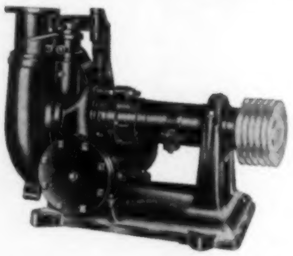
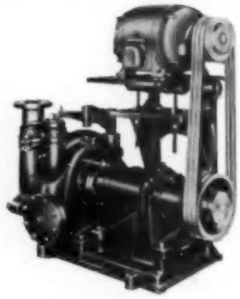

MINING
engineering
OCTOBER 1957

WILFLEY SAND PUMPS

LOW COST PUMPING



The new patented WILFLEY MODEL "K" centrifugal sand pumps are designed for maintained high efficiency in rugged service and trouble-free operation. They combine the original WILFLEY principles with new improvements developed through years of engineering experience.



BELT DRIVEN, overhead V-belt driven and direct driven WILFLEY Model "K" pumps available in 1", 1½", 2", 2½", 3", 4", 5", 6", and 8" discharge sizes with capacities to 3000 GPM and heads as high as 200'.

WILFLEY SAND PUMPS may be fitted with interchangeable electric furnace alloy iron or rubber covered wear parts to best meet the requirements of each pumping installation. The WILFLEY wear part design is ideal for rubber covering as ample clearance around the runner assures a practical and dependable rubber lined pump. Wherever you have a solids transfer job specify the WILFLEY Model "K" sand pump — for continuous, trouble-free performance, simple installation and stepped-up production.

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A. R. WILFLEY and SONS, INC.

DENVER, COLO., U.S.A.—NEW YORK OFFICE: 122 E. 42ND ST., NEW YORK CITY 17

Wilfley Sand Pumps
"Companions in Economical Operation"
Wilfley Acid Pumps

The AIME also publishes **Journal of Metals** and **Journal of Petroleum Technology**

Published monthly by the American Institute of Mining, Metallurgical, and Petroleum Engineers, Inc., 29 West 39th St., New York 18, N. Y. Telephone: PEnnsylvania 6-9220. Subscription $8 per year for non-AIME members in the U. S., & North, South, & Central America; $10 foreign; $6 for AIME members, or $4 additional in combination with a subscription to "Journal of Metals" or "Journal of Petroleum Technology". Single copies, $.75; single copies foreign, $1.00; special issues, $1.50. The AIME is not responsible for any statement made or opinion expressed in its publications. Registered cable address, AIME, New York. Indexed in Engineering Index, Industrial Arts Index, and by The National Research Bureau. Second class mail privileges authorized at New York, N. Y., and additional entry established at Manchester, N. H. Number of copies of this issue 15,700.

MINING engineering

VOL. 9 NO. 10 — OCTOBER 1957

COVER

Hydraulic mining—cover artist Herb McClure's unit operation this month—is a reminder of one of the Tampa meeting highlights, the phosphate production field trip. Latest Tampa program additions are reported on page 1153 and 1158.

Know Your Society

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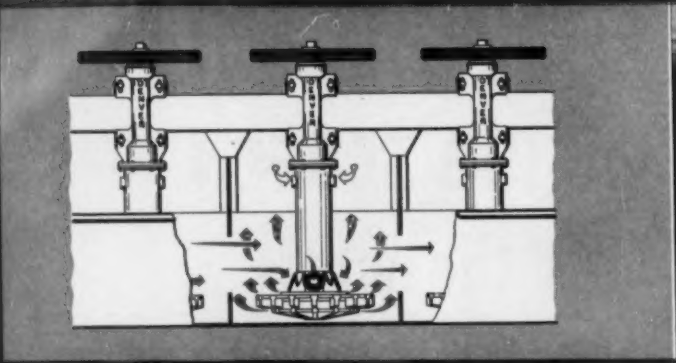
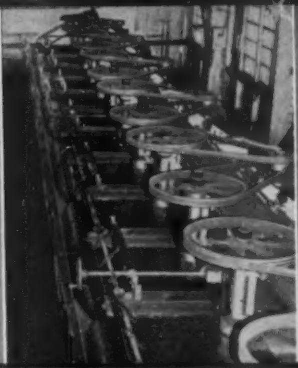
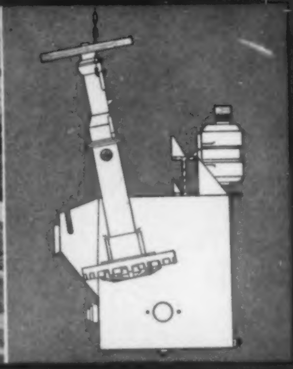


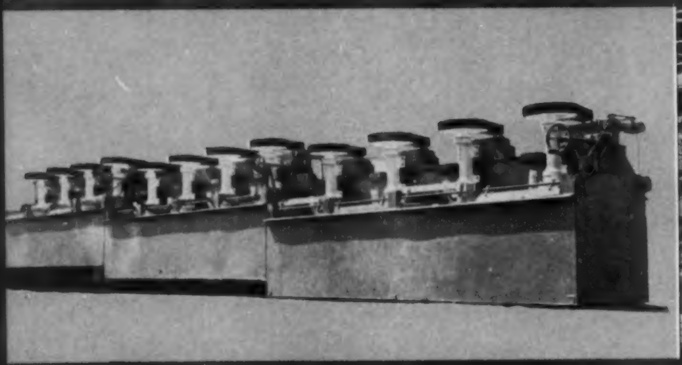
OPEN TYPE CELL DESIGN permits simplicity, low costs, high efficiency in flotation rougher circuits.



LESS PARTS—Longer service—18,231,000 tons in 12½ years. Maintenance less than ¼c per ton.



LESS DOWN TIME—SIMPLE Part Replacement



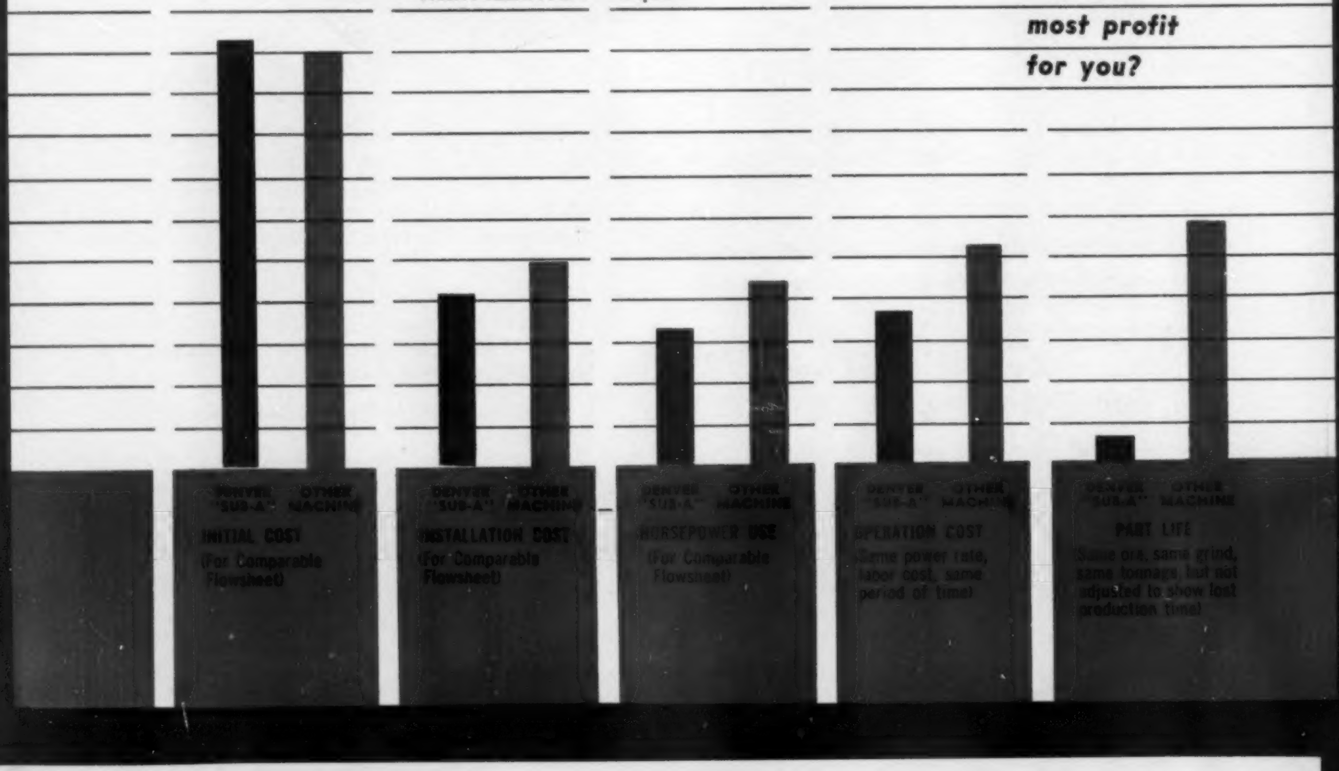
LESS COST—SIMPLE Machine, SIMPLE, Low-cost, High Tonnage Circuits

LESS POWER—High efficiency at Low-Horse Power. BIG then at high tonnage.

COMPARE SPECIFICATIONS, PRICE AND PERFORMANCE

METALLURGY—Equal

Which machine will make most profit for you?



DENVER "Sub-A" ROUGHER FLOTATION

means greater NET * PROFITS

* Denver "Sub-A" Flotation Machine has done what no other machine was able to do during the past 40 years.

RECENT CASE HISTORY—

At this plant various competitive flotation tests have been conducted to determine if there was a **better rougher flotation machine** than the existing plant machine. During the past three years this 1680 ton per day mill established two parallel identical 840 t.p.d. rougher flotation circuits. One circuit served as control, other circuit was used for tests. Competitive tests were conducted between (A) existing plant machine (control circuit), and (B) a Denver "Sub-A" Flotation Machine and (C) one of the other competitive rougher machines.

DENVER "Sub-A" Flotation Machines were purchased by this company for their rougher flotation circuit because:

INITIAL COST was competitive.

SAVINGS ON INSTALLATION COST due to simplified mill construction and fewer pumps.

SAVINGS ON HORSEPOWER of 30%.

SAVINGS ON OPERATION through simpler operation of machine and circuit.

LONGER PART LIFE — DENVER "Sub-A" parts lasted up to 4 times longer plus extra production due to less maintenance.

METALLURGY—Equal to slightly better.

—Don't overlook the unseen factors in flotation that affect your profits.

"The firm that makes its friends happier, healthier and wealthier"

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PERSONNEL

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service, Inc. (Agency) operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Geophysicist, 29, B.S. in geophysics. Four years mining exploration experience in Canada, Alaska, U. S., and South America, using all types of geophysical equipment. Speak fluent Spanish. Prefer foreign location. M-355-San Francisco.

Geologist, 31, A.B. and M.S. degrees. Three and one half years exploration and property evaluation experience and two years mine geology, development, and engineering experience in iron, nonferrous metals, and nonmetallics. Prefer eastern U. S. M-356.

Geologist, 42, B.S. and **M.S.** degrees. Experience in exploration and mining geology, metallic and nonmetallic mineral deposits of many types including placers. Considerable administrative experience. Now employed United Nations in Far East. Seeking position with growing company engaged in active exploration and development of diversified mineral deposits. Prefer western U. S. but will consider other locations. Available early 1958. M-357.

Mining Engineer, 1955 graduate, age 25, married, one child. Four years summer experience in Canadian mining. Completing two years Army service. Desire opportunity to regain experience and prove ability, U. S. or Canada. Available October first. M-358

Mining Engineer, B.S. and M.S. degrees, age 47. Eight years experience laboratory research and ore testing, eight years mineral search and appraisal on three continents, three years mining and two years engineering instruction. Owns patented gold prospect. Desires position in research or analysis, appraisal and report writing. Prefer U. S. or Canada. M-359.

Economic Geologist, M.S. degree, age 29. Six years experience geological mapping and exploration for metals and nonmetals in U. S and Latin America. Some geochemistry and core logging. Resident work preferred. Location desired, Latin America, south or east U. S. M-360-DM9325.

Mining Geologist, 30, B.S. and M.A. in geology, married, one child, World War II veteran. Four years experience in exploration, development, and underground mining. Prefer responsible position with good future. M-361.

Mine Superintendent, B.Sc. and E.M., age 46. Twenty years experience in mine operation and production work. Underground vein and flat-bedded metal mines. Up-to-date on mining methods, excellent personnel handling, safety, and cost control records. Employed, available 30 days. Prefer east or northeast U. S. M-362-832-Chicago.

POSITIONS OPEN

Junior Mining Engineer, for survey plan work. Must be good computer. Salary, open. Location, Florida. W5309 (a).

Engineers. a) *Mineral Dressing Engineer,* B.S. in mineral dressing, M.S. preferred, with two to five years experience in field. Will perform economic analyses of ore beneficiation processes. Excellent opportunity for growth into key position. Salary, to $9000 a year. c) *Ore Dressing Engineers,* B.S. in metallurgy or mineral dressing, with up to five years experience, to perform development work on ore beneficiation processes. Salary, to $7200 a year. Locations, a) Pennsylvania, c) Michigan W5295.

Department Head, research, both applied and basic, in mineral beneficiation; Ph.D. desirable but not necessary; preferably at least ten years experience in ore dressing and extractive metallurgy. Must be able to carry developments through to production usage. Salary, $11,000 to $13,000 a year. Location, Florida. W5189.

Engineers, for mining operation. a) *Mechanical-Electrical Engineer,* with experience in power generation and transmission motors, installation and maintenance, machine shop, etc., for gold lode property. Base salary, $7800 to $8400 a year; social benefits provided by local law bring salary up to approximately $9,420 to $10,080 a year. Furnished house provided free of charge. b) *Assistant Mill Superintendent,* with experience in cyanidation and lead flotation plants for 500-tpd gold mill. Base salary, $6000 a year; social benefits provided by local law bring salary up to approximately $7260 a year. Furnished house provided free of charge. Location, South America. F5163.

Geophysicist-Geologist, to 35, bachelor's degree in geology or phy-

(Continued on page 1046)

Going down...

down...

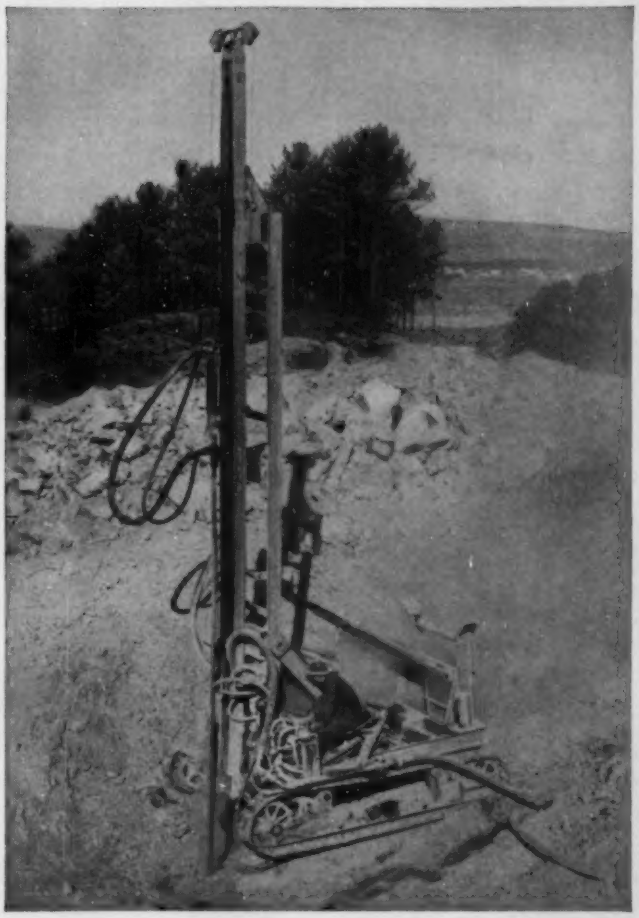
down...

with a

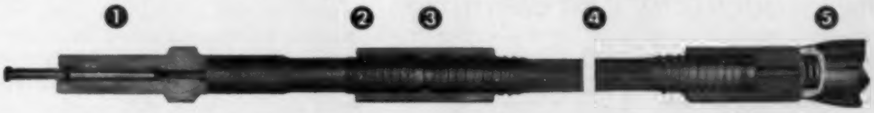
Gardner-Denver

DH143

Don't settle for halfway measures when you want rock drilling depth. Move in and drill deep holes fast with the Gardner-Denver DH143. This hard-hitting rock drill has a $5\frac{1}{2}''$ hammer diameter . . . handles rod changes to 20′ . . . delivers the heavy-duty, deep-hole punch needed in pits, quarries and open cuts. It allows you to select the proper bit and plan the proper spacing for the best breakage in any ground.



GARDNER-DENVER SECTIONAL DRILL ROD EQUIPMENT WITH RING SEAL SHANKS



Use Gardner-Denver ring seal shanks and sectional drill rods with the DH143. They are made like the highest quality rock drill parts. Special steels combined with the finest metallurgical techniques—shot peened and carburized—give this drill steel lasting deep hole drilling durability.

1. **Ring Seal Shank**—trouble-free, long-wearing "O"-ring seal.
2. **Threads**—specially long for greater durability.
3. **Couplings**—designed and developed for the job.
4. **Rods**—specially treated like the finest quality rock drill parts.
5. **Carbide Bit**—holds gauge on deepest holes.

ENGINEERING FORESIGHT—PROVED ON THE JOB
IN GENERAL INDUSTRY, CONSTRUCTION, PETROLEUM AND MINING

GARDNER - DENVER

Gardner-Denver Company, Quincy, Illinois
In Canada: Gardner-Denver Company (Canada), Ltd., 14 Curity Avenue, Toronto 16, Ontario

For roughing, scavenging and cleaning

Wemco Fagergren Flotation Machines

deliver the most per foot

and per dollar.

Where flotation machines are used by the acre, the Wemco Fagergren is by far the most frequent choice. The reasons are equally important to the user of many cells or few.
Because the large operators can afford and do run the most exhaustive competitive tests, they are able to determine which flotation machines provide the most profits at least cost.
Their choice of Wemco Fagergrens is based on these well proven advantages:

1. Greatest capacity per dollar investment
2. Highest recovery of values
3. Smallest floorspace requirements
4. Minimum reagent cost
5. Lowest supervision requirement

Wemco's field engineers can show you the proof.

Write today for further information

WEMCO®

WESTERN MACHINERY CO.

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Representatives throughout the United States and Canada and in major countries around the world.

Here's why the JR-38B

UNIVERSAL JACKDRILL

**gives you
MAXIMUM FOOTAGE
PER SHIFT
at
LOWEST
OVERALL COST**

FEATURES

- Roll-Type Feed-Leg Valve built into handle
- Five-Position Throttle controls all functions
- Built-in Air Connection between drill and leg
- Adjustable Tension on hinge joint
- Adjustable Balance for easy handling
- Enclosed Steel Centralizer
- Constant Chuck Blowing
- One-Piece Chuck with renewable bushing
- Extra-Rigid Fronthead Construction
- Lighter Weight Telescopic Leg

Set-up time can be converted to drilling time with the Ingersoll-Rand JR-38B Jackdrill – the easiest-handling one-man universal drilling unit ever developed. Designed to take full advantage of the inherent economies of 1⅜" Carset Jackbits, it saves drilling time, too, and reduces air requirements as well. The JR-38B is rugged – it stays on the job. These savings add up to maximum footage per shift and a new low in drilling costs.

The features at the left give some of the reasons for this outstanding Jackdrill performance. For the complete story, see your I-R man or send for a copy of Bulletin 4144A.

CARSET JACKBIT

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IR

DRIFTERS · JACKDRILLS · JACKHAMERS · CRAWL-IR DRILLS · CARSET BITS · AIR TOOLS · COMPRESSORS

Personnel

(*Continued from page 1042*)

sics, with minimum of five years practical experience as qualified geophysicist. Should be capable of planning and conducting preliminary preparations for geophysical work on various projects; evaluate reliability of results and make geological interpretations of seismic data, coordinate and evaluate gravimeter and magnetometer work in relation to reflection seismic work. Salary, to $10,200 a year. Single status preferred; one-year family separation if married. Location, Far East. F5157.

Administrative Assistant, degree in mining engineering, 27 to 32, with three to six years experience in mining, to prepare ore and metal studies considering economic as well as technical aspects; review reports on *outside* properties to determine potential as mining ventures and summarize important facts; review monthly progress reports from subsidiary companies, etc. Salary, $6000 to $9000 a year. Location, New York. W5131.

Field Geologist, for large construction firm operating in Peru. One to two years experience in the field. Single status. Salary, $7200 a year plus room and board. Three year contract. F5124.

Research Engineer, mineral beneficiation, 24 to 35, M.S. degree in mineral beneficiation and up to two years experience, or B.S. in mineral beneficiation and one to three years experience. Experience preferred in research or development facilities with broad interests, i.e., not specializing in one phase of mineral beneficiation. Experience in operations of a mineral beneficiation plant processing nonmetallics acceptable. Salary, open. Location, West Coast. W-5112-C6319.

BOOKS

•ORDER BOOKS SO MARKED THROUGH AIME—

Address Irene K. Sharp, Book Dept. Ten percent discount given whenever possible.

Metal Statistics—1957, published by the *American Metal Market*, 18 Cliff St., New York 38, N. Y., 856 pp., $3.50, 1957.—The *little red book*, as it is known in the metal industries, contains over 400 statistical tables covering production, consumption, prices, and other data on ferrous and nonferrous metals.

Engineering Analysis, A Survey of Numerical Procedures, by Stephen H. Crandall, *McGraw-Hill Book Co. Inc.*, 417 pp., $9.50, 1957.—Three broad classifications of problems are recognized: equilibrium, eigenvalue, and propagation. Three items frequently omitted in engineering courses are developed: matrix notation, the calculus of variations, and the theory of characteristics of partial differential equations. •

The Detroit Industry Engineering Committee, ECPD, is offering six booklets available from the *Engineering Society of Detroit*, 100 Farnsworth Ave., Detroit 2, Mich., at $2.50 per set. The booklets cover general introductory material, orientation and training in industry, appraisal and counseling on performance, continued education, reading list for engineers, and professional identification.

ASTM Proceedings, *American Society for Testing Materials*, 1916 Race St., Philadelphia 3, Pa., 1510 pp., $12.00, 1957.—The 1956 edition contains reports of 80 technical committees with appendices and 52 technical papers with discussions. Many of these reports were given at the Society's national meetings. •

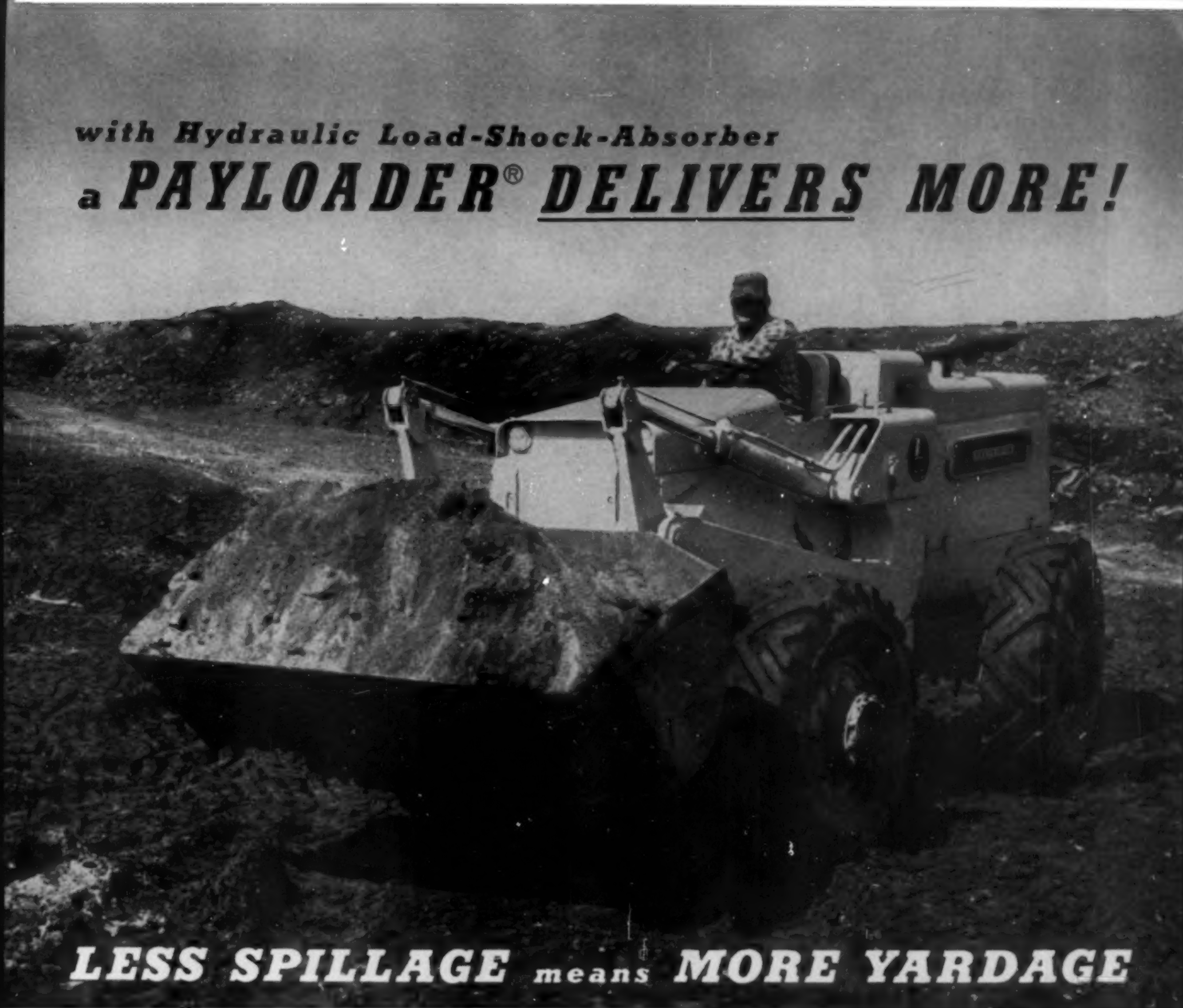
Solvent Extraction in Analytical Chemistry, George H. Morrison and Henry Freiser, *John Wiley & Sons Inc.*, 269 pp., $6.75, 1957.—This book, the first of its kind, offers comprehensive coverage of trace concentrations, the first section being devoted to principles and the second to the practical aspects, including apparatus and techniques. •

Proceedings of the 1957 National Industrial Research Conference, *Armour Research Foundation of Illinois Institute of Technology*, 35 W. 33 St., Chicago 16, Ill., $6.00, 1957.—The proceedings include 12 papers on management level decisions on research and development which were presented by top executives at the conference in Chicago in April. The papers deal with sales growth through research.

Iron Ore Beneficiation, by Lawrence A. Roe, *Minerals Publishing Co.*, P.O. Box 85, Lake Bluff, Ill., 305 pp., $5, 1957.—A unique book devoted to the tremendous development in beneficiation processes as applied to iron ores, this publication offers a wide range of detailed information under headings such as flotation, magnetic separation, agglomeration, etc. One of the most interesting sections is that on sources and economics which provides historical background on the influence of beneficiation on iron ore and taconite production. One of the book's greatest values lies in the extensive collection of references to recent periodical and technical literature particularly on material which has never or not yet appeared in any of the standard reference handbooks. •

Supervisors Safety Manual, *The National Safety Council*, 425 N. Michigan Ave., Chicago 11, Illinois, 354

(*Continued on page 1171*)



It's the yardage you *deliver* that counts — not how much you *dig*. The difference is the spillage that occurs between digging and dumping points — the amount you handle for nothing.

"PAYLOADER" tractor-shovels are designed to *deliver* more yardage — to dig big loads and to move them with the least spillage loss — because "PAYLOADER" and only "PAYLOADER" among wheeled tractor-shovels has hydraulic load-shock-absorber as standard equipment — that cushions the loaded bucket, eliminates bucket jounce, smooths the ride, and permits higher carrying speeds with less spillage. Other "PAYLOADER" design features that reduce spillage losses are the longer wheelbase and the low, close and stable load-carry position with bucket in full 40° tip-back just off the ground.

You get more performance from a "PAYLOADER" becausee you get more tractor-shovel . . . power-transfer differentials, no-stop power-shift transmission, planetary final drives, power-steer, 4-wheel power-brakes . . . closed, pressure-controlled hydraulic system . . . powerful pry-out digging action.

Your "PAYLOADER" Distributor is ready to prove that a "PAYLOADER" can out-perform anything in its class — to have you try one on *your* work and let *you* be the judge. Call him today.

PAYLOADER®

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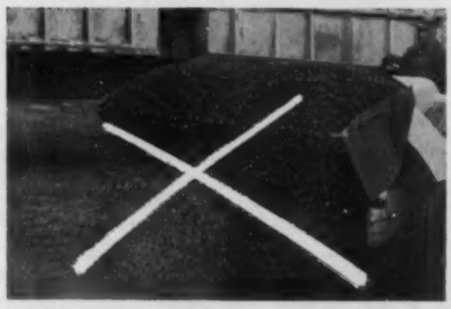
THE FRANK G. HOUGH CO. LIBERTYVILLE, ILL.

SUBSIDIARY—INTERNATIONAL HARVESTER COMPANY



THE FRANK G. HOUGH CO.

916 Sunnyside Ave., Libertyville, Ill.

Send full data on 4-wheel-drive tractor-shovels as checked:

☐ Model HO 2¼ yd. payload 1¾ yd. struck
☐ Model HH 1¾ yd. payload 1½ yd. struck
☐ Model HU 1½ yd. payload 1 yd. struck

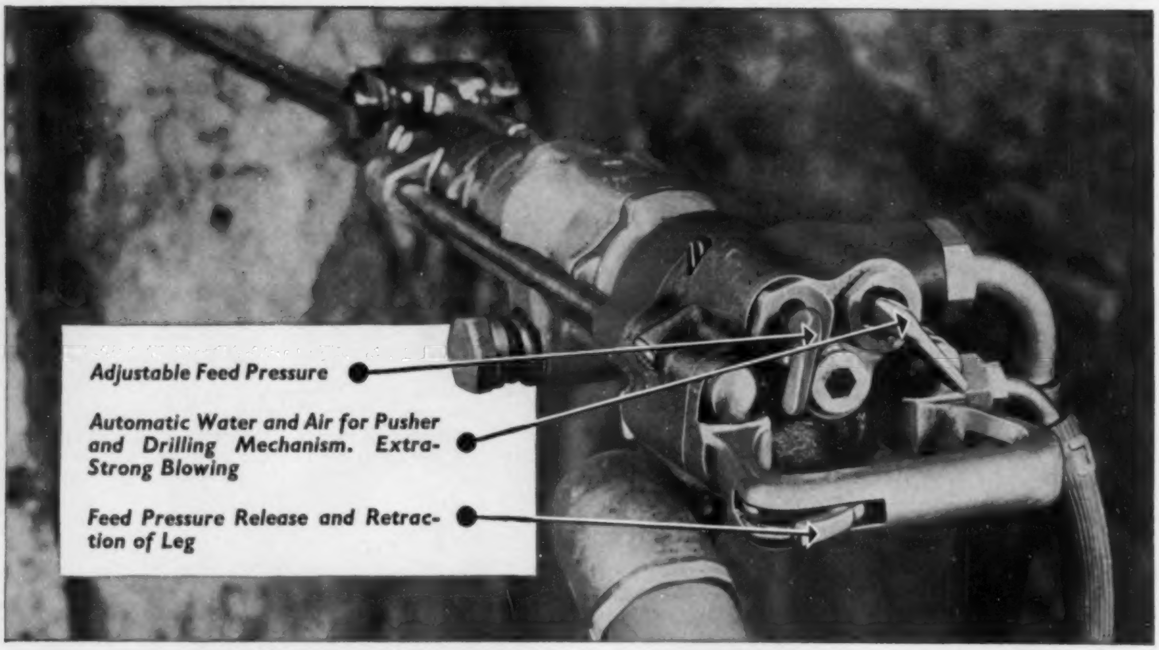
Name

Title

Company

Street

City State



THE ATLAS COPCO LION – A REVOLUTIONARY NEW ROCK DRILL

All controls under one hand

The Atlas Copco Lion is the first drill to have all the valves which operate the drill under the control of one hand. *Full and easy control without having to move the hand from the backhead!* All the controls have been designed so that they are well protected. While using them the operator's hand is never near the wall or roof of the drift. The Lion is the first pusher leg drill with controls placed for drifting.

Retractable leg saves time

When the leg has to be moved the feed pressure is easily released by squeezing the hand grip. By further pressure on the grip the leg retracts automatically. When the leg is in the new position suitable for continuous drilling, retraction stops and the feed pressure comes back by loosening the grip of the hand. *All this can be done while the drill is still running.*

This new idea of a retractable leg enables quicker repositioning of the leg and reduces the number of steel changes, thereby increasing footage per manshift. When drilling high holes it is now far easier to alter the position of the leg more frequently in order to maintain an optimal feed angle and feed pressure.

Packed with power for deep holes

The Lion has a drilling rate at least 30% higher than other rock drills of the same weight. Furthermore, it can maintain its high speed *even when drilling deep holes.* This means quickly drilled deep hole rounds and a faster advance. The Lion also reduces *to a minimum* the gauge wear of the bits in abrasive rock. And owing to the ease with which the feed pressure is released and brought back into action, the Lion is a *handier* drill to work with in fissured rock.

Sandvik Coromant—the steel for the Lion

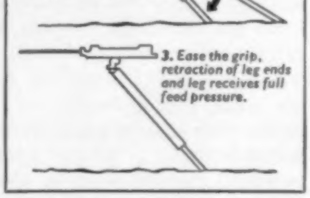
All Atlas Copco drills—and this goes for the Lion too—have been developed from the earliest stages with Sandvik Coromant tungsten-carbide-tipped integral steels and detachable bits. This combination pioneered tungsten carbide drilling in the early forties. No drill or steel developed separately could ever give such equivalently high performances as this drilling combination. Today it is the most widely used in the world, responsible for drilling more than 1,000 million feet annually.

For further information on the Atlas Copco Lion rock drill, and details of sales and service, please contact:

U.S.—Atlas Copco Pacific, Inc., 930 Brittan Avenue, San Carlos, California, or Atlas Copco Eastern Inc., P.O. Box 2568, Paterson 25, N.J.

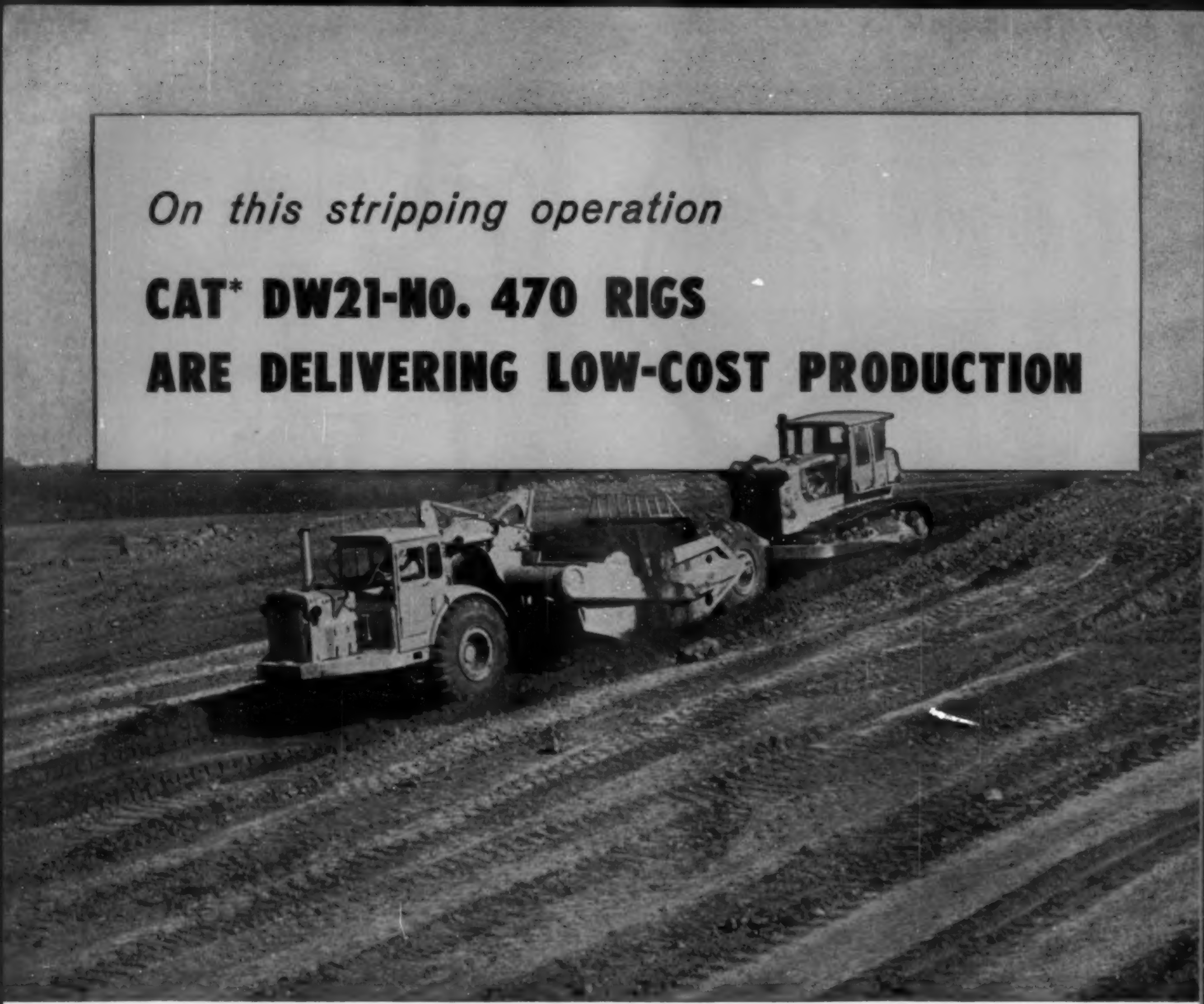
CANADA—Atlas Copco Canada Limited, Montreal Airport, P.Q.

MEXICO—Atlas Copco Mexicana S.A., Apartado Postal 56, Torreon, Coahuila.



Atlas Copco

Manufacturers of Stationary and Portable Compressors, Rock-Drilling Equipment, Loaders, Pneumatic Tools and Paint-Spraying Equipment



You see plenty of rugged Caterpillar-built machines at work on the Marquette Cement Company's stripping operation at Oglesby, Ill. In fact, you don't see any other make. That's the result of years of profitable experience, Superintendent W. T. Spurr will tell you.

The picture above shows one of the firm's seven big DW21s with No. 470 LOWBOWL Scrapers being push-loaded by a D9 Tractor. And it shows something else, too—how to move material at the lowest cost per yard.

These DW21-No. 470 rigs have capacities for big-volume hauling—18 cu. yd. struck, 25 cu. yd. heaped. But more important than capacity, the No. 470's LOWBOWL design has greater loading efficiency than *any other make* of scraper in its class. Actual tests show that the No. 470 gets its rated capacity load and is on its way while other scrapers are still in the cut struggling for the last few yards.

And when it has its big load, the DW21 moves it at high speeds, efficiently. Its 300 HP (maximum output) Cat-built Engine has a Turbocharger to add horsepower at reduced fuel costs. Air is packed into the engine *according to engine load,* not speed. Wide-section, tubeless, 29.5-29 tires provide excellent flotation and furnish greater tractive effort.

Let your Caterpillar Dealer show you how these rugged rigs can move material faster and cut costs on your operation. You can depend on him for sound advice now—for fast service and parts you can trust after you buy.

Caterpillar Tractor Co., Peoria, Illinois, U. S. A.

CATERPILLAR*

WANTED—
THE HARD WORK

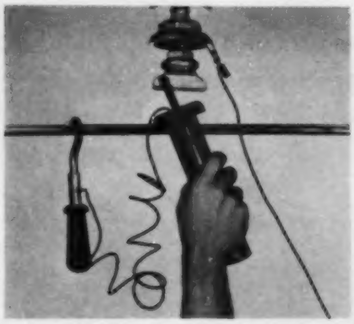
Manufacturers News

News
Equipment
Catalogs

• FILL OUT THE CARD FOR MORE INFORMATION •

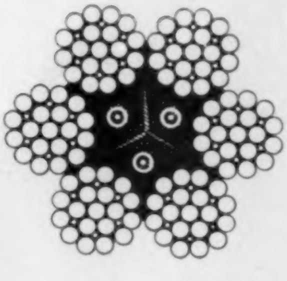
Insulated Hanger Tester

Ohio Brass Co. offers a simple unit for testing insulated mine hangers on the line or in the shop. Assembly consists of tester unit, rubber handled contact hook, and a ground wire. A glowing light in the tester case indicates defective insulation. **Circle No. 2.**



Signal Carrier Wire Rope

SignalKore wire rope extends its function as a hoisting cable by imbedded copper wires that permit its use in transmitting low voltage current for voice or signal communication. The rope, by Wire Rope Divs., *American Chain & Cable Co. Inc.*, is



shown above in cross-section with three-conductor core. Constant contact with a mine skip or automatic operation of a cage by the operator is possible with the new development. **Circle No. 1.**

Curing Power Loss

Reduction of horsepower caused by thin air at high altitudes can be eliminated on the D8 tractor by a normalizing kit available from *Caterpillar Tractor Co.* Kit consists of a turbocharger, manifolds, and adaptors. Turbocharger delivers air to the inlet manifold at sufficient pressure to restore normal horsepower; it does not increase power of the tractor at sea level. **Circle No. 3.**

Diaphragm Pump

Denver Eqpt. Co. announces production of a new 4-in. adjustable stroke diaphragm pump. Capacities to 70 gpm are offered for simplex model and to 140 gpm for duplex at ¾ to 1 hp, depending on specific gravity of pulp. Unit is only 50 in. high. Acid-proof construction is available. **Circle No. 4.**

Vibrating Hopper

Simplicity Engineering Co. has designed its new heavy-duty vibrating hopper for rapid loading of skip hoists. Hopper, measuring 6 ft square at top, is mounted on a main frame with 14 coil springs to isolate vibration. Tandem units may be mounted

on a common frame for use with twin skip hoists. Used this way, the units are claimed capable of emptying 15 tons of moist ore into skips in an average of 5 sec. **Circle No. 5.**

Cable Fault Locator

A pocket-size unit from *Ohio Brass Co.* fills the need for an inexpensive and easily used device for locating short circuits and open circuits in mine trailing cables having two or

more conductors. Total unit weighs 4 lb and is suitable for testing cables up to 600 ft long. Six-volt battery and signal generator are attached to end of cable and left there; only headphones and receiver are carried by the operator. **Circle No. 6.**

Crawler Drill Rig

Another model has been added to the *Gardner-Denver Co.* line of rock drills. The Air Trac crawler-mounted deluxe drilling rig provides power positioning of all vertical, horizontal, and flat lifter holes. Equipped with a 4 or 4½-in. drill, the rig is powered by two 5-cyl radial air motors. **Circle No. 7.**

Six-Yard Tractor Shovel

The rig shown below heaps a 14-cu yd hauler in two passes in less than 2 min, according to *Clark Eqpt. Co.* Shovel is a 6-yd Michigan 375A with

375-hp engine. It has a top highway speed of 30 mph. Torque converter automatically matches engine speed to load demand. **Circle No. 8.**

New & Notes

The Swedish mining machinery maker A. B. Nyhammar Bruk has been granted exclusive authority by *Lake Shore Inc.* to manufacture and sell the JETO bottom dump skip in Sweden, Norway, Denmark, and Finland.

The only REVERSIBLE cap and adapter made...

Try this on-for SAVINGS!

Service-proved WEARPACT Cap Type Tooth —with foolproof retainer—cuts digging costs to record lows . . . even in taconite!

Probably by now you've heard of Wearpact Steel—the alloy with a combination of hardness and toughness not available in any other commonly used alloy.

Here's a new tooth and adapter design* cast in longer-wearing Wearpact . . . the ASF Cap Type Tooth and adapter —the *only* such tooth and adapter that's reversible for longer life. Note that the tooth itself is self-sharpening. Keeps on *slicing* through the ore and overburden—and when finally worn, there's less metal to throw away. *The tooth locks securely to the adapter.* No ASF retainer has ever failed in service.

As for service life, the cost-per-ton figures in large mines speak for themselves. Lowest of any tooth on record! Give this dollar-saving tooth a try! Write today to arrange for an installation.

Cut your costs with WEARPACT

CAST ALLOY STEEL

A PRODUCT OF

AMERICAN STEEL FOUNDRIES

3750 Canal St., Indiana Harbor, Indiana

*Patent Pending.

(21) ELECTRICAL CONDUCTOR CHART: All past and present code designations for aluminum electrical conductors are brought together for the first time on a reference wall chart now available from *Kaiser Aluminum & Chemical Sales Inc.* Listed alphabetically are more than 600 code names with type and size of products specified. British, as well as U. S., standard tables are included on this large plastic paper chart.

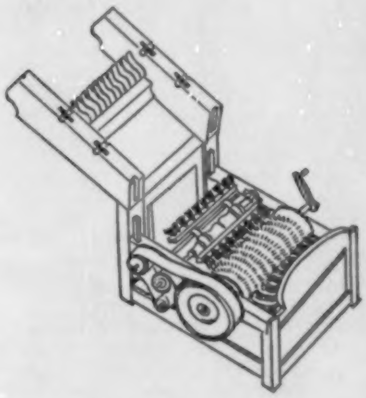
(22) TRANSMISSION PRODUCTS: An 80-page catalog of transmission products and their uses in elevating and conveying machinery has been published by the *Jeffrey Mfg. Co.* Descriptive information with drawings and tables covers a variety of shaft collars, couplings, clutches, pillow blocks, take-ups, wheel hubs, gears, holdbacks, chains, and sprocket wheels. The products catalogued are also adaptable to many applications in the industrial machinery field.

(23) OFFSHORE PLATFORMS: A 6-page folder with 20 illustrations has been prepared by *R. G. LeTourneau Inc.* to describe its line of self-contained heavy-duty offshore platforms. The brochure discusses in detail the size, capacities, and operational versatility of one specific unit, the Vinegarroon. Of particular interest are illustrations which include a drawing of deck layout, below deck facilities, progressive ocean-floor illustrations of platform going on location. Additional photographs show construction phases, living facilities, deck cranes and other nondrilling equipment on board.

(24) STATIONARY DIESELS: Bulletin 110 from White Diesel Engine Div., *White Motor Co.*, describes the Models 60 and 80 Superior stationary diesels. Engines are four-cycle, 6 or 8-cyl, in-line models, ranging 530 to 2000 bhp.

Free Literature

(25) ELI WHITNEY: Believing that the work of Eli Whitney deserves to be more widely recognized, the *Bennett Machinery Co.* has gathered some of the highlights of his career and set them forth in a booklet,



"The Founder of Modern Industry." The brochure discusses Whitney as the first American machine tool builder and the founder of modern methods of manufacture, in addition to his well known invention of the cotton gin.

(26) MINE MACHINERY: "Modern Mining," a catalog of modern, heavy-duty mine machinery, has recently been released by *Caterpillar Tractor Co.* Showing how the crawlers, wheel tractors, tractor-shovels and motor graders help increase production, the brochure gives actual job reports. The booklet also lists condensed data on some of the more recent units.

(27) WELDING GUIDE: Fifth edition of a 40-page "Guide to Better Welding" is offered by *Marquette Mfg. Co.* Included is a trouble chart, table of steel hardness numbers, and a welding rod comparison chart.

(28) TIN RESEARCH: A booklet from the *Tin Research Institute* covers 25 years of work of the International Tin Research Council. Principally a review of institute progress, the booklet also contains a synopsis of some new uses for the ancient metal.

(29) ROCK DRILLING: *Le Roi Div.*, Westinghouse Air Brake Co., has a new 8-page bulletin, "I Know Airpower," which explains remedies to troubles encountered when using rock drills. Brochure also outlines features of the Le Roi Newmatic air tools and equipment.

(30) MINING TOOLS: *Firth Sterling Inc.* announces a catalog of mining tools including mining machine bits, roof bits, drill bits, finger bits, and drill bit inserts.

(31) SPECIAL METALS & ALLOYS: Metallurgists and design engineers who are concerned with heat resistant alloys, abrasion resistant materials, and with new alloys for extremely severe processing conditions will be interested in the new *Shieldalloy Corp.* brochure on pure metals, master alloys and ferro alloys. Just published, this booklet covers various grades of chromium metal, chromium molybdenum, ferro alloys, tungsten melting base alloy, zirconium aluminum and vanadium aluminum. The Shieldalloy brochure also lists a number of experimental alloys which have recently been developed at their laboratories and which are available in laboratory and pilot plant quantities. For all of these alloys the brochure lists complete chemical analysis, as well as powder mesh sizes.

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

10

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after Dec. 15, 1957—if mailed in U. S. or Canada

Please send { More Information ☐ / Price Data ☐ / Free Literature ☐ } on items circled.

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Company————————————————

Street——————————————————

City and Zone———————————State————

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
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41	42	43	44	45	46	47	48	49	50
51	52	53	54	55	56	57	58	59	60
61	62	63	64	Students should write direct to manufacturer.					

(32) PNEUMATIC CONVEYING: Bulletin No. 143-B, issued by *Spencer Turbine Co.*, describes both stationary and portable pneumatic conveying systems. Systems cover a capacity range from ¾ to 10 tph, vacuum producers from 5 to 75 hp, and hose or pipe sizes from 2 to 7 in. Conveying under both pressure and vacuum is covered and several typical applications are illustrated. Specific information is provided on size of unit, rate of conveying, type and weight of material in each case.

(33) WIRE ROPE ABRASION: Leschen Wire Rope Division, *H. K. Porter Co.*, has released new Bulletin No. 101 which describes the principal causes of abrasion, the commonest and most destructive enemy of wire rope, and lists numerous practical suggestions for minimizing its effects. This is the first of a new series of bulletins which will provide information on common wire rope problems and their best solution.

(34) DRILL BITS: A new general catalogue covering "Oriented Diamond" drill bits has just been released. Especially designed for quick, easy location of important drill bit data, the catalogue by *Hoffman Bros. Drilling Co.* lists sizes, setting charges, weights, and uses for all standard bits. New price sheets are also being distributed.

(35) ARC WELDING: Revised by *Ampco Metal Inc.* to include data on the new Ampco-Trode 40 electrode and new Ampco Braz #1, #2 and #3 filler rods, Bulletin W-17 contains up-to-date technical information relative to arc welding with bronze electrodes, filler rod and wire. Described in the catalog are welding procedures, techniques and processes, recommended welding currents, typical applications and a comprehensive electrode and filler rod selection and preheat chart.

(36) ORE INVESTIGATION: A new 10-page illustrated bulletin has just been prepared by *Southwestern Engineering Co.* to describe the services of its ore investigation laboratory. Of particular interest is the presentation of modern methods of ore concentration, including flotation, magnetic separation, chemical processing, pyrometallurgical and gravity methods.

(37) AIRSTREAM CONVEYORS: A new 32-page product bulletin, "Airstream Conveyors—the Automatic Answer to Bulk Handling," has just been published by *Dracco Corp.* This bulletin presents detailed technical information on the pneumatic conveying solution to many bulk materials handling problems. System diagrams of typical installations point out the applications and flexibility of these conveyors. Over 70 illustrations are included to show how air transports granular or powdered bulk material efficiently. The bulletin lists materials which can be handled and describes specially-engineered systems.

(38) FRICTION: "Friction Facts," a new booklet that answers questions about friction material in clutch facings and brake linings, has been released by *Caterpillar Tractor Co.* This booklet points out vital characteristics friction material must have in order to withstand the severe conditions imposed by earthmoving machines. Loss of friction, glazing and chipping, wear and torque capacity are discussed.

(39) BRASS HOSE FITTINGS: A new 8-page bulletin on brass hose fittings for low and medium pressure service is available from *Hose Accessories Co.*

(40) HIGH TENSION BELTS: *Boston Woven Hose & Rubber Co.* offers a revised folder of specifications on its line of high tension belts.

(41) CONSTRUCTION MAINTENANCE: A new 32-page special construction maintenance issue of the semi-monthly Technical Information Digest (TIS 2832) is now available from *Eutectic Welding Alloys Corp.* Devoted entirely to industry's construction maintenance problems, this special issue is designed to show welding maintenance staffs the way to practical savings through case histories. Techniques are explored for every job done in construction fleet maintenance. Particular emphasis is placed on repair of cast iron, manganese and carbon steel, metals most commonly used in hard working construction equipment. The economy of preventative maintenance through hard surfacing is also treated.

(42) HARD-FACING ALLOYS: A new data sheet (No. 5), listing grinding wheel recommendations for finishing overlays of Colmonoy Hard-facing alloys, is now available from *Wall Colmonoy Corp.* The new data sheet lists in tabular form recommended wheels for use on Colmonoy nickel-base, cobalt-base and iron-base alloys. Part numbers are given for wheels of the following manufactures: Atlantic, Bay State, Blanchard, Carborundum, Cortland, Maklin, Norton, Precision, Robertson, Simonds, and Sterling. Recommendations are offered for grinders of the following types: surface (both horizontal and vertical), cylindrical, internal, centerless, and tool room.

(43) GEAR HANDBOOK: A 1957 handbook from *Ohio Gear Co.* is packed with 186 pages of power transmission data compiled and revised to meet today's modern engineering standards and tabulations. The gear section of the new handbook has been expanded and tabulates complete dimensions on Ohio Standard spur, bevel, spiral, helical, worm gears, and steel rack. Speed reducers are included.

(44) SAFETY EQUIPMENT: *General Scientific Equipment Co.* has just published a new catalog presenting the entire company line of safety equipment in an easy to use buyer's-guide arrangement. The catalog includes all types of protective equipment, from a small half-ounce dust mask to a large lifter. Nearly all of the several hundred listed items are illustrated and general specifications and recommended uses are given for each. Equipment is grouped according to its function so that the reader can compare similar products.

(45) EXPLOSIVES BAGS: Laminated burlap, multiwall paper, and polyethylene plastic bags are among the specialized bags offered in a new booklet from *Chase Bag Co.* Included is the laminated cemented center seam, gusseted Protex bag, of interest to users of ammonium nitrate.

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MINING ENGINEERING

29 WEST 39th STREET

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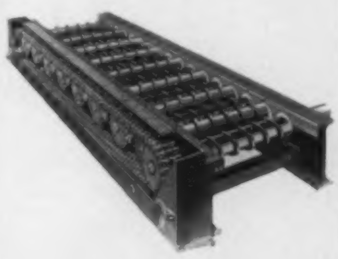


World's Largest Shovel is the Mountaineer, built by Marion Power Shovel Co. for Hanna Coal Co. It is 16 stories high, and scoops up overburden in 90-ton bites. USS "T-1" steel is specified for such major components as the bucket, dipper stick and crowd rack; to assure the strength, toughness and ruggedness that adds up to long service life.

"T-1"... a super-tough nickel alloyed steel prevents breakage... slashes downtime

At 38°F below zero...or at 900°F above..."T-1" steel stays *super-tough!*

Downtime troubles *go out* when "T-1" steel *goes in* dipper sticks, shovel buckets, car bottoms, bulldozers and other mining equipment.

That happens because "T-1" steel is a constructional alloy steel combining exceptionally high levels of strength and toughness. It stands up under heavy shock loads. Curbs breakage the year 'round.

Note its advantages:

90,000 psi minimum yield strength.

Readily weldable . . . needs no preheat nor stress relief.

Resists impact at sub-zero temperatures.

Four times as resistant as carbon steel to atmospheric corrosion.

Use "T-1" steel for heavily stressed parts. Watch them stay on the job, earning profits. Why take chances with steel of lower yield strength and consequent risk of failure in service? Idle men and machines make it a costly proposition...so, put "T-1" steel to work now.

For details on USS "T-1" steel, write to United States Steel Corp., Pittsburgh 30, Pa.

INCO TRADE MARK **THE INTERNATIONAL NICKEL COMPANY, INC.** 67 Wall Street New York 5, N. Y.

PRINTED IN U.S.A.

Five Uranium Firms Plan Merger

Atlas Corp. has announced that, on approval of stockholders, three Atlas-controlled companies and two independent firms will merge to form what will be the largest independent U. S. uranium mining company. Carried through, the negotiation will form a company having proven reserves with a value in place of over $100 million. Atlas firms involved are Hidden Splendor Mining Co., Lisbon Uranium Corp., and Rio de Oro Uranium Mines Inc.; the independents are Mountain Mesa Uranium Co. and Radorock Resources Inc. Reasons for the merger: stability, strength, operating efficiency. New firm will use the name Hidden Splendor Mining Co.

Freeport Arranges for Moa Bay Financing

Freeport Sulphur Co., through its wholly owned subsidiary, the Cuban American Nickel Co., has arranged for loans amounting to $100.25 million to cover the major share of funds need to exploit the nickel and cobalt ores of Moa Bay, Cuba, and to construct a refinery near New Orleans. Production is slated for mid-1959. Freeport expects the Moa Bay area to add 50 million lb of nickel to U. S. supply, as well as 4.4 million lb of cobalt. Estimated nickel reserves in Cuba, the company claims, are about six times larger than those of Canada.

Erecting Test Plant for Oregon Lateritic Nickel

New Delhi Mines Ltd. is conducting an extensive sampling program on its lateritic nickel deposits in Josephine County, Oregon, and will build a test plant to evaluate the more than 15 million tons of ore which have been indicated by churn drilling during the past few months. Development of new processes in treating laterite deposits and success of the Nicaro, Cuba, nickel production from laterite deposits has drawn the attention of steel and alloy producers to the Oregon operations. Orebodies are about 60 miles from Hanna Nickel Smelting, which is currently producing ferro-nickel under Government contract.

Homestake Seeks Public Financing

For the first time, Homestake Mining Co. will look to financing in the public market. It recently registered with the Securities and Exchange Commission issues amounting to $12 million of sinking fund and subordinate convertible debentures. A possible tie with this offering is Homestake's partnership with Sabre Pinon Corp. to mine uranium in New Mexico. An agreement which resulted in the formation of Homestake-Sapin Partners includes provision for a $17 million mill construction and mining program. Homestake has advanced $7.5 million and has arranged for the remainder through bank credit.

Plan Largest Integrated Titanium, Zirconium Producer

Three metals companies intend to form a jointly owned concern which will produce titanium and zirconium from ore to finished mill product. Intended producer, which is to be the largest fully integrated one in the field, will be formed by P. R. Mallory & Co., Sharon Steel Corp., and National Distillers & Chemical Corp. Plans call for Mallory-Sharon Titanium Corp., which is jointly owned by Mallory and Sharon, to acquire the production facilities of National Distillers and make a change in name to Mallory-Sharon Metals Corp. Acquisition would also be made of Reactive Metals Inc., a research and development company which is jointly owned by Mallory-Sharon and National Distillers.

After drilling 15 to 20 shoot-holes, the Tractair moves to the next job. "We save time and labor because the Le Roi unit doesn't need to be babied into position," says Green. "We drive the Tractair onto rocks, slopes, or loose ground and start drilling."

Tractairs* Save $150 a Day at Blasting Site

Le Roi tractor-compressor used to drill shoot-holes for blasting overburden

Easier and safer to handle on rough terrain than dozer-drawn compressors, superintendent says

The V. N. Green Co., Brown's Branch, W. Va., saves $50 per day at each site by using Le Roi Tractairs to drill shoot-holes for blasting surface rock at their strip mining operations. When all three Tractairs are used, as is generally the case, daily savings total $150, reports John W. Green, job superintendent.

Previously-used wheel type compressors had to be moved around the site by a standby dozer. The 10 to 15 moves per day took two hours. The dozer lost the same amount of time, too, when its work was interrupted.

Mobility Minimizes Delays

Green points out that now "the Tractair is driven directly to the drilling area, the hose is attached, and drilling starts at once." In addition, Tractair's easy maneuverability over the extremely rough terrain cuts previous relocation time by more than half, and eliminates the need for the dozer as well as the operator.

Because it's self-propelled, the Le Roi tractor-compressor is ideal for this type of work. When a job is done, the Tractair provides fast transportation for the operator, drill, and compressor to the next site. There's no need to wait for other equipment to make the move.

What's more, close-to-the-job air power shortens hose lines, substantially reduces hose damage, and eliminates power loss. The shorter lines and Tractair's 125 cfm output permit the use of two drills, when necessary.

Performs Various Jobs

Bought originally for drilling shoot-holes, the Tractair can be used for other jobs, too. "We also use our Tractairs to break paving on road construction jobs where we do the resurfacing, to tamp backfill for drain pipes, and even to clean pavement by using a rotary brush attachment."

The self-propelled, multi-purpose Tractair can provide similar savings for you. And its ready adaptability to a wide range of attachments can help you reduce the cost of operations requiring costly, one-purpose equipment. Contact Le Roi today.

*"Tractair" is the registered trademark for Le Roi's combination tractor—air compressor.

WABCO

LE ROI Division of Westinghouse Air Brake Co., Milwaukee 1, Wisconsin, manufacturers of Newmatic air tools, Tractair,® portable and stationary air compressors, and heavy-duty industrial engines. Write us for information on any of these products.

V-71

Boron Minerals

New Keys to Higher Fuel Energy

Borax has made a dramatic jump from washtub to jet bomber with the development of a process to manufacture high energy fuels that use boron as a base. Olin Mathieson Chemical Corp., which recently opened the first semi-commercial plant to produce the new fuels, expects that a $1-billion industry based on these high energy agents will emerge within the next ten years. Olin Mathieson also estimates that production of high energy fuels will increase 20-fold within an 18-month period.

Need for special fuels arises from the requirements of high speed aircraft for a tremendous rate of heat release in a light weight fuel. HEF fuels are regarded as those with heating values well above those of petroleum products, such as aviation gasoline, which has a heating value of about 18,500 Btu per lb. High energy fuels range 25,000 Btu and up.

While the major steps of the fuel making process remain clouded for security reasons, it is known that the manufacture of HEF fuels consists of treating boron-containing ores in such a way that they are converted to useful intermediate boron compounds. Subsequent reactions then convert these into the final product.

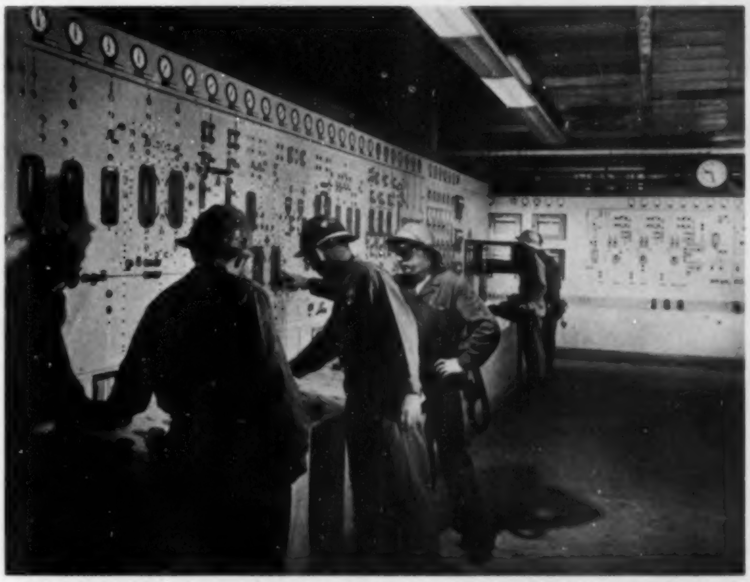
Why Boron?

Liquid hydrogen, on the basis of energy content alone, would be the ultimate pure energy fuel. But it is elusive, difficult to hold and to handle. It must be contained and its burning rate must be controlled. This is more easily done chemically than mechanically. But when a fuel is burned, the chemical carrier is consumed at the same time as the hydrogen. When the goal is fuel of maximum heating value, it is necessary to find a carrier material that has both a high heating value in itself and is capable of holding as much hydrogen as possible in relation to its own weight. Measured this way, nature's chemical binder carbon is not outstanding. It has a heating value in the pure state of only about 13,000 Btu per lb.

Scientists took the logical tack of working downward from hydrogen in the list of heating values of the elements. Next to hydrogen on the list is the metal beryllium, which surrenders about 29,000 Btu per lb when burned. But beryllium has the disadvantage of being relatively scarce and difficult to recover and refine.

Boron, next on the list at 26,000 Btu per lb, is far more encouraging. In the form of borax and colemanite, boron is plentiful in California and other parts of the West.

Lithium, next at 18,000 Btu, although also fairly accessible, was so much lower in heating value that it was decided to focus the greatest effort on boron. The problem was then one of linking boron and hydrogen, and possible other elements, in proportions giving optimum energy, safety, and economy.



Control of all operations of Olin Mathieson's high energy fuel plant is centered in this room. Complex panels are considered small in relation to those that will be used in in a $36 million Air Force plant which will start producing in 1959. The industry founded on these boron-based fuels is expected to reach billion-dollar value in the next ten years.

New Plant

After five years of research and two years of production testing, Olin Mathieson found the answers. The fuels have been designated HEF-2 and HEF-3. These new agents are now being produced in a semi-commercial plant built solely by the company at a cost of $5.5 million. An interim facility, scheduled for completion at the end of 1957, will turn out three times as much fuel. All the fuel, which at the present time is considered suitable for aircraft only, is shipped to the U. S. Air Force. It will be used in air-breathing engines. A USAF-sponsored plant, now under construction by Olin Mathieson, is expected to go into production within 20 months and will increase output many times current volume. Its construction cost will be $36 million.

Boron Mineral Sources

Three companies supply 95 pct of the Free World's borax needs. U. S. Borax & Chemical Corp. supplies 70 pct of domestic production from its California mines, the largest in the world. A unique deposit of rasorite (kernite), recently converted from underground to open pit operation, is worked by its Pacific Coast Borax Div. American Potash & Chemical Corp. produces 20 pct of domestic output by extracting borax from the dry crust of Searles Lake near Death Valley. The West End Div. of Stauffer Chemical Corp. recovers borax from the rich brine deposits of Searles Lake and accounts for 10 pct of domestic production.

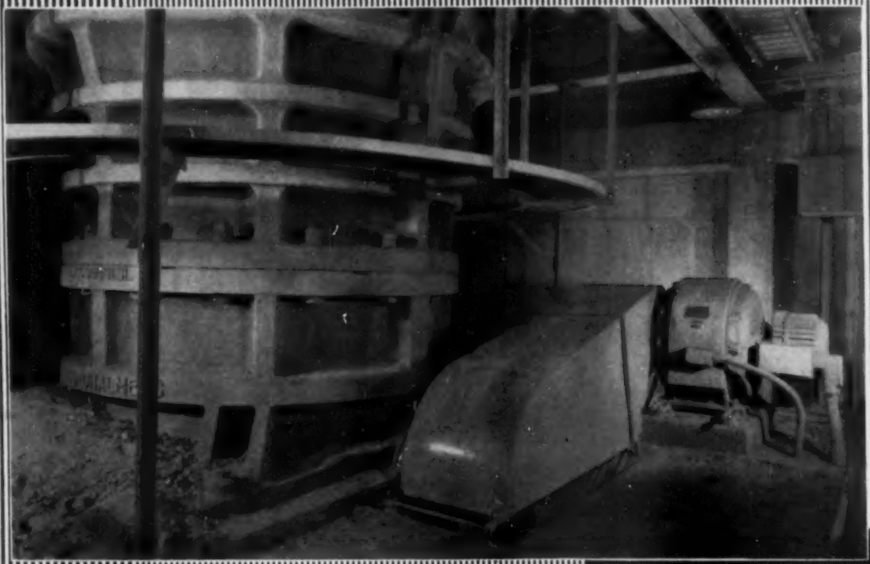
Last year sales of boron minerals amounted to 944,950 tons valued at almost $40 million, according to the Bureau of Mines. Five years ago sales were valued at just over $14 million.

How "COORDINEERING" at

. . . Better

"Coordineering" may be defined as the integration of advanced thinking, planning and engineering that precedes the application of Allis-Chalmers equipment. With its staff, skill and experience, Allis-Chalmers is in a unique position to provide this valuable service.

Allis-Chalmers designs and builds all basic types



Crushing

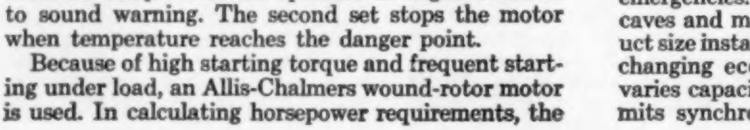
An example of Allis-Chalmers "coordineering" is this crusher installation. Drawing on 75 years' experience in building and applying crushers, Allis-Chalmers control and crushing engineers developed a control circuit designed to cut costly downtime for the crusher installation shown above. In this circuit, motor overload protection is provided by *two* sets of thermal overload relays. One set operates at slight overload to sound warning. The second set stops the motor when temperature reaches the danger point.

Because of high starting torque and frequent starting under load, an Allis-Chalmers wound-rotor motor is used. In calculating horsepower requirements, the factors of crushability, ratio of reduction, product size and specific gravity are evaluated.

Flexibility in the Crusher

The crusher is the *Superior* gyratory crusher which features "one-man, one-minute" positioning of mainshaft and mantle. This control facilitates emptying crushing chamber in case of power failure or other emergencies. It also compensates for wear on concaves and mantle and, when required, changes product size instantly. In Allis-Chalmers gyratory crushers, changing eccentricity, speed, or shape of chamber varies capacity and product size. This flexibility permits synchronizing crushing with other operations.

Superior is an Allis-Chalmers trademark.



Hammermills   Vibrating Screens   Jaw and Gyratory Crushers   Grinding Mills   Kilns, Coolers, Dryers

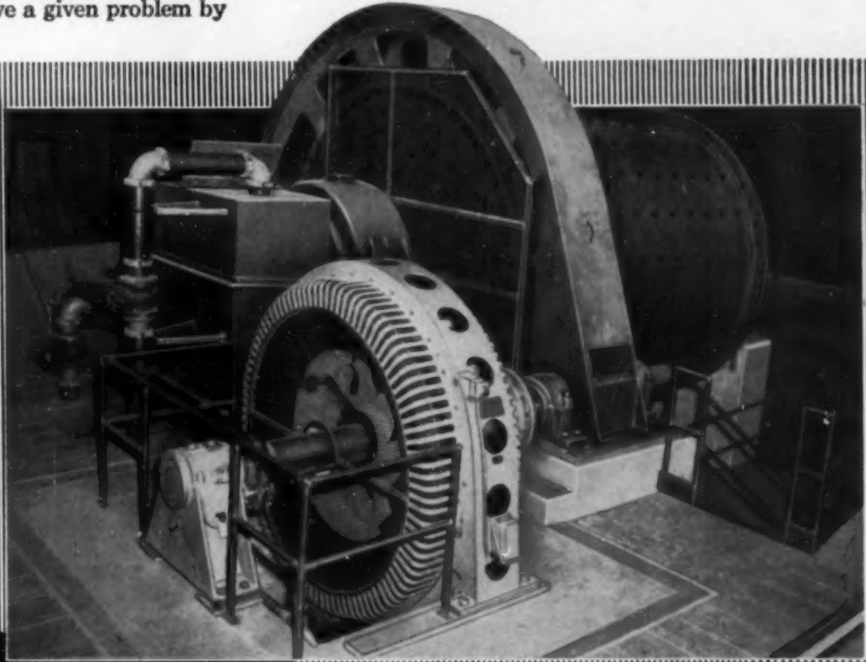
ALLIS-

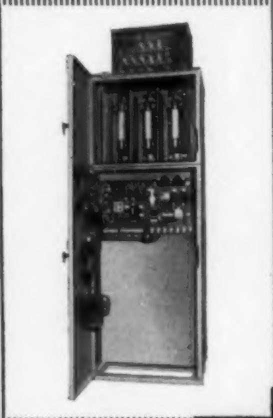
Allis-Chalmers Provides Better Methods

Results for You!

of processing machinery. In addition, the company manufactures complete lines of electrical generation, distribution and utilization equipment. As a result, Allis-Chalmers has a tremendous reservoir of experience—a most diversified team of research, engineering, manufacturing and application specialists — specialists who solve a given problem by exchanging ideas and correlating specific know-how and skills.

Only Allis-Chalmers can give you truly integrated equipment — because only Allis-Chalmers can give you "coordineering."





Grinding

In solving a grinding application, requirements and variables are given a careful going over by an Allis-Chalmers team comprised of *grinding*, *motor* and *control* engineers. Characteristics of material, capacity, feed preparation, balance of gradations, torque characteristics, system power factor needs, control requirements are some of the many factors evaluated. Experience has proved that this thorough pre-application investigation and preparation pays off to the purchaser . . . *pays off in providing a modern, efficient grinding circuit with the lowest operating cost.*

An Integrated Grinding Mill Installation

In a typical application, the mill installed was a 10½ by 12-foot Allis-Chalmers diaphragm ball mill. The close diameter-length ratio is an important factor in producing the highest possible capacity per unit of power. Driving the mill is a 900-hp, 4000-volt, 257-rpm, 0.8 pf Allis-Chalmers synchronous motor. By providing desired power factor correction, this motor reduces power cost. The Allis-Chalmers starter is specially engineered to provide protection under all conditions of grinding operation.

A-4915

★ **You'll want Bulletin 25C6166D.** It covers all equipment manufactured by Allis-Chalmers for the mining industries. See your nearby A-C representative or write Allis-Chalmers, Industrial Equipment Division, Milwaukee 1, Wisconsin.

CHALMERS

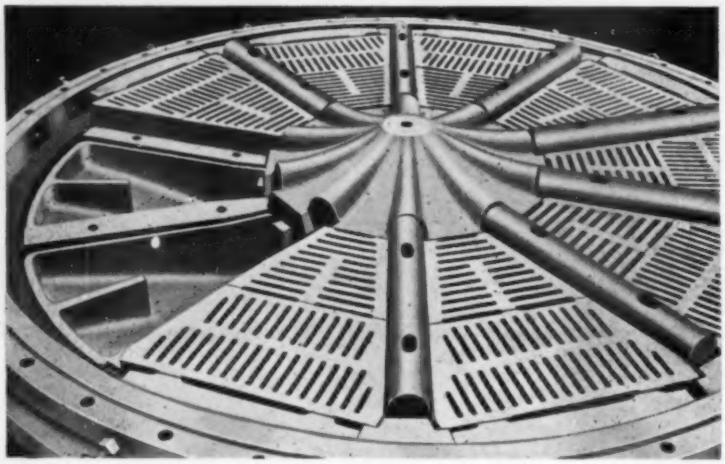
A-C ALLIS-CHALMERS

initial or operating cost?

see how Marcy can give YOU long-time economy

Low production costs are becoming more and more important for profitable operation. Continuous, dependable performance of milling equipment is essential for low costs...but, low initial equipment cost does not assure low-cost-per-ton production.

The predominance of Marcy Mills in large companies having big tonnage and continuous operation is evidence of Marcy's superior performance...repeat orders from these companies prove the preference for Marcy design and construction for low-cost-per-ton production.



***"Rapid Change of mill content** is necessary for high efficiency."* That's the Marcy principle of grinding. It is accomplished by use of *full-grate* discharge on Marcy Ball Mills and the *open-end* feature on Marcy Rod Mills. It has proved to increase tonnage up to 33% and decrease KWH per ton, compared with overflow type mills...one reason why Marcy gives lower cost-per-ton and makes a better long-time investment.

NO EXTRA CHARGE FOR MARCY® EXPERIENCE

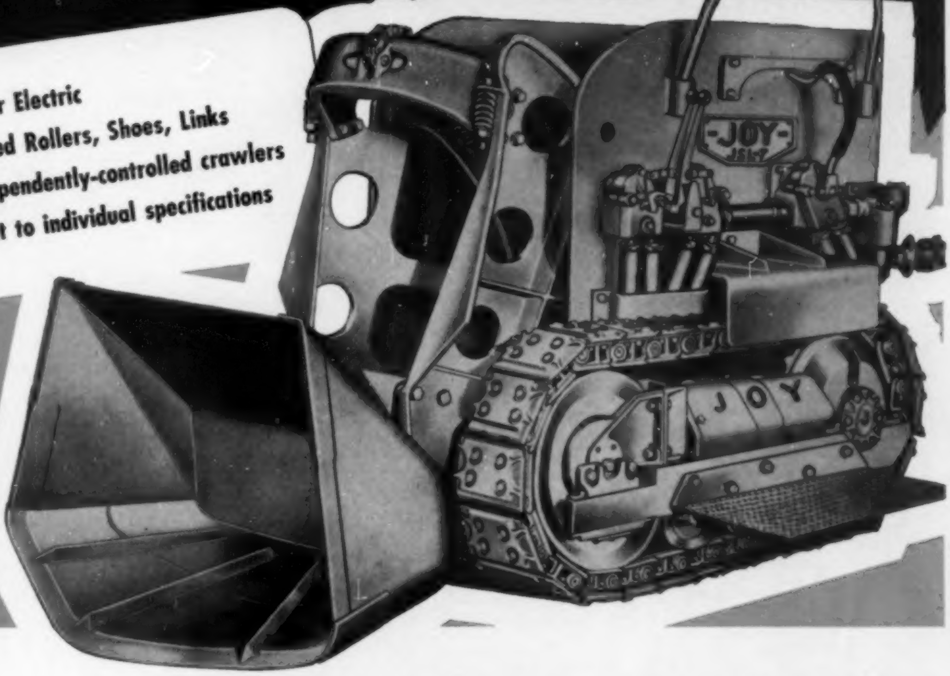
THE MINE AND SMELTER SUPPLY CO.

DENVER NEW YORK SALT LAKE CITY EL PASO

SALES AGENTS AND LICENSED MANUFACTURERS THROUGHOUT THE WORLD

New! JOY HARD ROCK LOADER

- Air or Electric
- Forged Rollers, Shoes, Links
- Independently-controlled crawlers
- Built to individual specifications



Joy JSL-7 Loader	Specs
Height, minimum	...60"
Headroom Range	79" to114"
Width, overall	...57½"
w/platform	68"
Length, bucket in digging position	...112½"
Weight, Air	9,650 lbs.
Electric	...11,250 lbs.
Loading, average tons per minute	...2—4 tpm

The all-new Joy JSL-7 Shovel Loader is designed and built specifically for the really rough hard rock jobs. This crawler-mounted brute has been beefed up with forged parts at all the tough impact points; has three interchangeable Pistonair® motors in the air version—(255 ft. lb. stall torque motors in the electric)—and delivers a 12,080 lb. crowding effort at the bucket lip.

Independent Crawler Control

Each crawler, independently powered by a famous Joy 15 hp Pistonair motor, is reversible ... the Joy JSL-7 can turn in its own length. No need for run-backs and repositioning when this Shovel Loader swings into action!

Control Positioning Optional

Standard positioning of controls on left side of Loader as pictured. Right-side mounted or dual mounted controls are also available in either air or electric models. "Deadman" controls flip to neutral when released.

Joy Builds to your Requirements

In addition to standard specifications tabulated at left, the Joy JSL-7 Shovel Loader is available with special buckets, arms and counter weights for mucking on grades greater than 9° uphill or 15° downhill. Arrangements for bucket discharge heights to 87" are available.

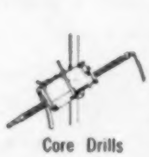
Call in the Joy Engineer and tell him what you want the JSL-7 to do for you. Ask him about the choices of bucket design and capacity, grousers, 'dozer attachments and other available extra equipment. Write to *Joy Manufacturing Company, Oliver Building, Pittsburgh 22, Pa.* In Canada *Joy Manufacturing Company (Limited), Galt, Ontario.*

WRITE FOR FREE BULLETIN 203-7

Write for this new bulletin covering the Joy JSL-7 Shovel Loader. 8 Pages of dimensions, weights and drawings.

WSW M8860-203

JOY ... EQUIPMENT FOR MINING ... FOR ALL INDUSTRY



Core Drills

Slushers

Rock Bits

Drillmobiles

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THE PORPHYRY COPPERS IN 1956

By A. B. Parsons, formerly Secretary, AIME

A new volume, supplementing the author's "The Porphyry Coppers" published in 1933, long out of print. Sponsored by the Rocky Mountain Fund of the AIME.

Summarizing the last 25 years of progress at the 19 Porphyry Copper properties in North and South America, the last seven of which were not in the previous volume:

Utah Copper	*Chino*	*Consolidated Coppermines*
Morenci	*Inspiration*	*Castle Dome*
Nevada Con.	*Chuquicamata*	*Copper Cities*
Braden	*New Cornelia*	*Bagdad*
Miami	*Copper Queen*	*Yerington*
Ray	*Andes*	*Silver Bell*
		San Manuel

288 pages: 31 photographs, 38 maps and drawings, 27 tables of data

Production, Sales, Income and Expense, Costs, Wage Rates, Financing, Government Contracts, Dividends; Equipment, Ore Reserves, Grades of Ore and Concentrate, Recoveries, Flowsheets, Drilling, Mining, Concentrating, Leaching, Smelting, Refining, Transportation.

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Send me promptly on publication *The Porphyry Coppers in 1956:*

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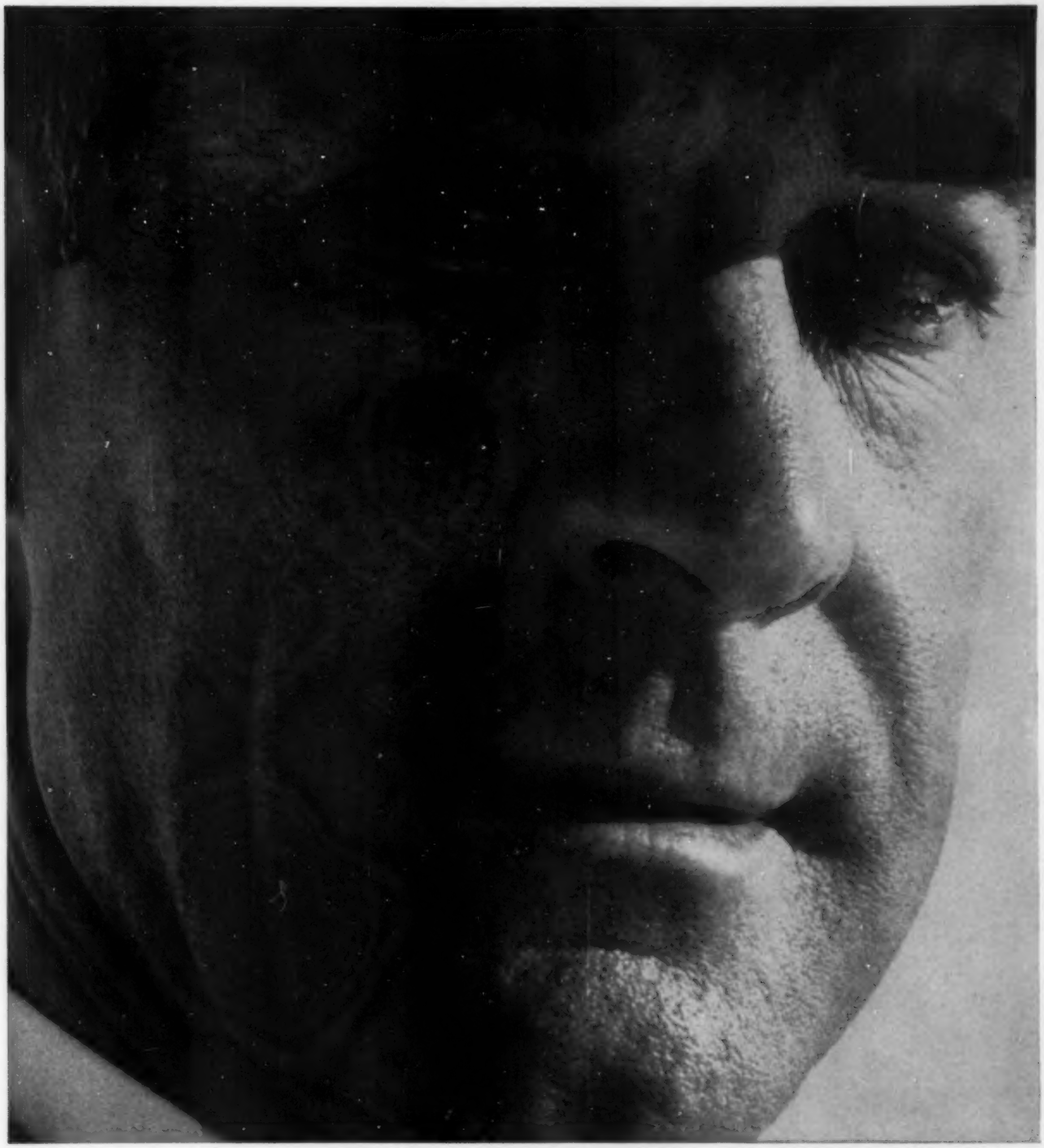
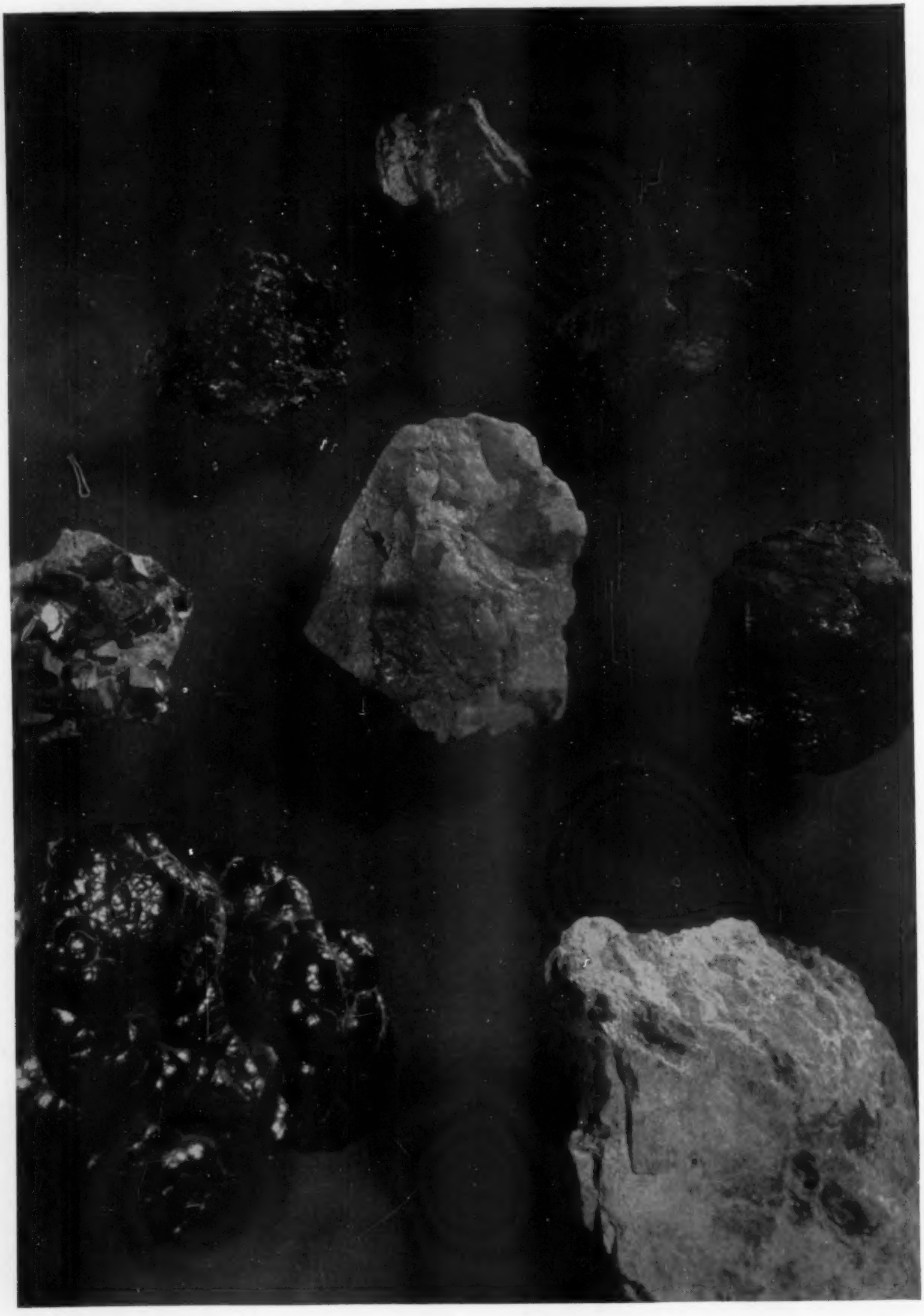
Mention of more than 260 individuals who have played important roles in the story.

A Must for Mining and Metallurgical Engineers, Mineral Economists, Investors.

Price only $3.50, postpaid, to AIME members anywhere. Others, $5.00, plus 50¢ postage to foreign countries.





These represent a few of the minerals with which Separan 2610 is improving flocculation and lowering processing costs: CHROMITE, URANINITE, GALENA, SPHALARITE, CHALCOCITE, ALUMITE, COAL, HEMATITE.

For a wide variety of minerals Separan 2610 provides record-speed flocculation

NEW FLOCCULANT PROVES ITSELF IN NUMEROUS OPERATIONS

It's a fact—proven in many types of operations, with a wide variety of materials: Separan 2610® has doubled, and often tripled, production rates!

Equally remarkable: only a few hundredths of a pound are needed per ton of solids!

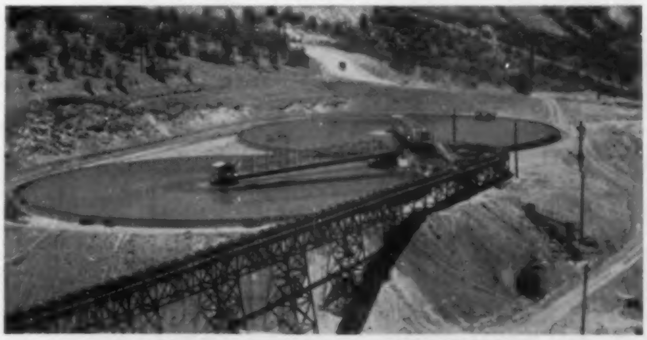
New economies are being realized in processing uranium, lead, and zinc, coal, copper, alum, and a host of other sulfide and non-sulfide minerals. Operations involving concentration, leaching, thickening, and refining are benefiting with Separan 2610, especially since this revolutionary flocculant is so easy to prepare and apply. A mechanical mixer is not required. You can prepare large amounts of stock solution with a new dispenser recently made available. And Separan 2610 is effective in both acid and alkaline circuits. In addition, it adds no excess bulk.

Try Separan 2610 yourself!

Available from three f.o.b. points: Midland, Michigan; Pittsburg, California; and Velasco, Texas . . . and from two stock points: Port Newark, New Jersey, and St. Louis, Missouri. A sample and technical assistance are yours upon request. THE DOW CHEMICAL COMPANY, Technical Service and Development, Midland, Michigan, Dept. SE1315B-1.

When your process calls for chemicals, talk to the man from Dow

XANTHATES • DOWFROTH* 250 • Z*-200
SEPARAN 2610 • HYDROCHLORIC ACID
AMMONIA • CHLORINE • CAUSTIC SODA
VERSENE* • MAGNESIUM OXIDE
MAGNESIUM HYDROXIDE • ION EXCHANGE
RESINS AND SOLVENT EXTRACTANTS.



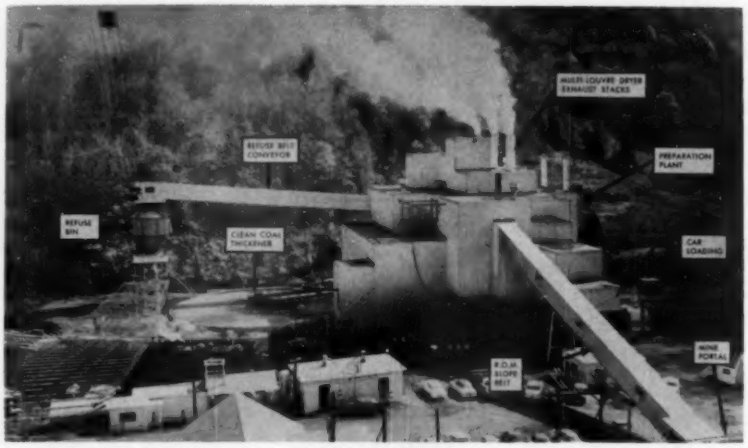
HERE ARE EXAMPLES of how little Separan 2610 is required: In thickening copper tailings, clearer overheads and better filtration are reported with only 0.0041 pounds per ton of solids. Similar improvements are taking place in uranium filtration at 0.5 pounds per ton.

YOU CAN DEPEND ON DOW

Two Men Control New Coal Plant

Until 1954, Moss No. 2 was an abandoned coal mine that had not been lit by a miner's lamp since the end of World War I. That year the mine in Clinchfield, Va., was explored and found to have a reserve of metallurgical coal, now much in demand.

Clinchfield Coal Co., owner of the property, reopened the old diggings after deciding that the move would be economical if an ultra-modern plant were built to process the coal. An automatic preparation facility was designed and erected by Link-Belt Co. using float-sink concentrators to quickly separate out rock, shale, bone, and other refuse.



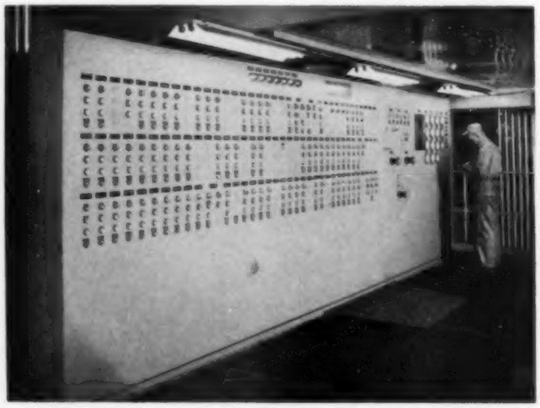
One of the latest, most modern processing plants in the coal industry, this facility serves the Moss No. 2 mine at Clinchfield, Va., and typifies the growing mechanization in coal production. It washes, dries, and screens 450 tons of run-of-mine coal an hour—by push-button control.

Plant Control

Operating two shifts a day, the plant handles 450 tph of run-of-mine coal. The entire operation is automatically controlled, with push-button handling from two stations. These control all washing, drying, and screening needed to produce wanted grades of metallurgical and steam coal.

All maintenance is performed during operating shifts. This necessitates a plant with dual set-ups, standby equipment, and a freight elevator for handling supplies.

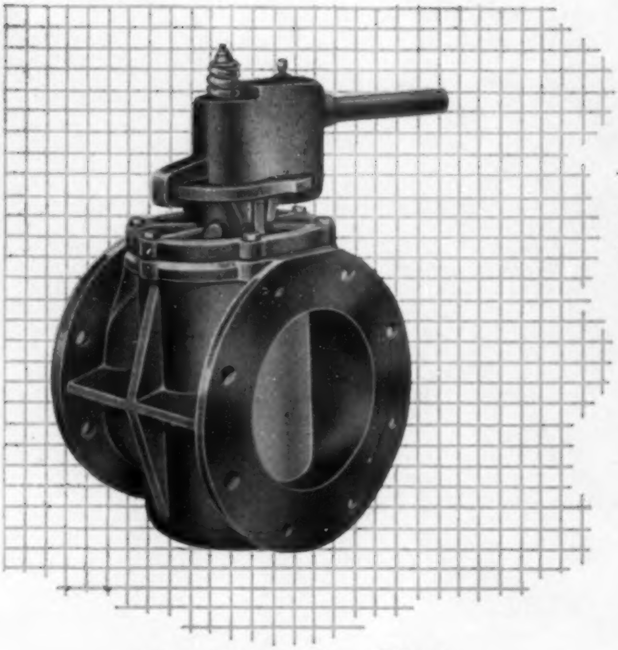
The facilities include heavy media cleaning for coarse coal; 20 concentrating tables for cleaning fine coal; four centrifugal and three thermal fine coal driers; a 95-ft circular thickener and filter for water circulation.



This one-man operated main control panel provides regulation of wet cleaning and processing of both coarse and fine coal. Another such panel, also operated by one man, serves the drier section.

Cleaning Operation

Clinchfield mixes finely ground magnetite solids with water to produce media with accurately controlled specific gravity. Raw coal is fed into concentrator vessels containing this material. The clean coal, being of lower specific gravity than the refuse, floats and is discharged over an outlet weir. Waste material sinks to the bottom of the concentrator and is removed to a flume by rotary elevators. The media is reclaimed by sieving, and the magnetite which clings to the coal is washed off and reclaimed by magnetic separators.



Designed for

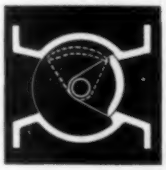
EASIER OPERATION..

TIGHTEST SHUT-OFF...

LONGER LIFE!

DeZurik Valves have these Exclusive Features

ECCENTRIC ACTION

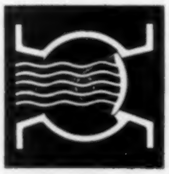


backs the plug away from the seat—no scraping or binding, no galling or grinding—easy quarter-turn from wide open to dead-shut.

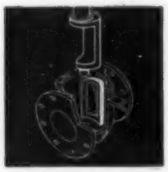
RESILIENT PLUG FACINGS

insure tight shut-off—even with solids in the flow! Sandy slurries *won't* score the plug-face—can't cause leakage and valve failure!

TIGHT SHUT-OFF



is guaranteed. The resilient-faced plug seals *around* solids in the flow, delivering perfect performance every time on any line!



NO LUBRICATION

is required with DeZurik Plug Valves. Eccentric action assures easy operation—resilient plug facings deliver tight shut-off—WITHOUT LUBRICATION!

DeZurik Plug Valves are the logical choice for so many mining applications. On hard-to-handle slurries, they shut tight and last longer; on critical services there is no danger of contamination from lubricants; and on any service they operate easily!

These versatile DeZurik Valves have features found on NO OTHER VALVE! Only DeZurik Valves have Eccentric Action, combined with resilient plug facings, guaranteeing easy operation and dead-tight shut-off WITHOUT LUBRICATION!

Before you specify new or replacement valves on that difficult service, get the complete story on DeZurik Valves. Representatives in all principal cities; or write

DeZURIK CORPORATION
SARTELL, MINNESOTA

A Bucyrus-Erie 150-B shovel strips wasterock and dirt in a copper pit in western United States. Two more Bucyrus-Erie electric excavators and a Bucyrus-Erie rotary drill are in the background.

WHEN IT'S HIGH OUTPUT YOU WANT IT'S BUCYRUS-ERIE YOU NEED

Many copper mines are helping protect their profit margins with Bucyrus-Erie Ward Leonard electric shovels. Sustained high output is essential to economical production . . . and consistently high production is yours with these Bucyrus-Eries.

Check their design and construction advantages and you'll see why they are known as top producers in mines everywhere. Outstanding front-end design keeps a high percentage of "up-front" weight *inside* the dipper where it pays. The two-piece boom is strong yet light in weight. Crowd machinery is mounted on the revolving frame, not on the boom. Result: power swings payload, not deadweight.

When selecting stripping and loading shovels, if you consider output, cost, and long life, you will want to consider these three great Bucyrus-Erie Ward Leonard electric excavators.

128L57C

BUCYRUS ERIE

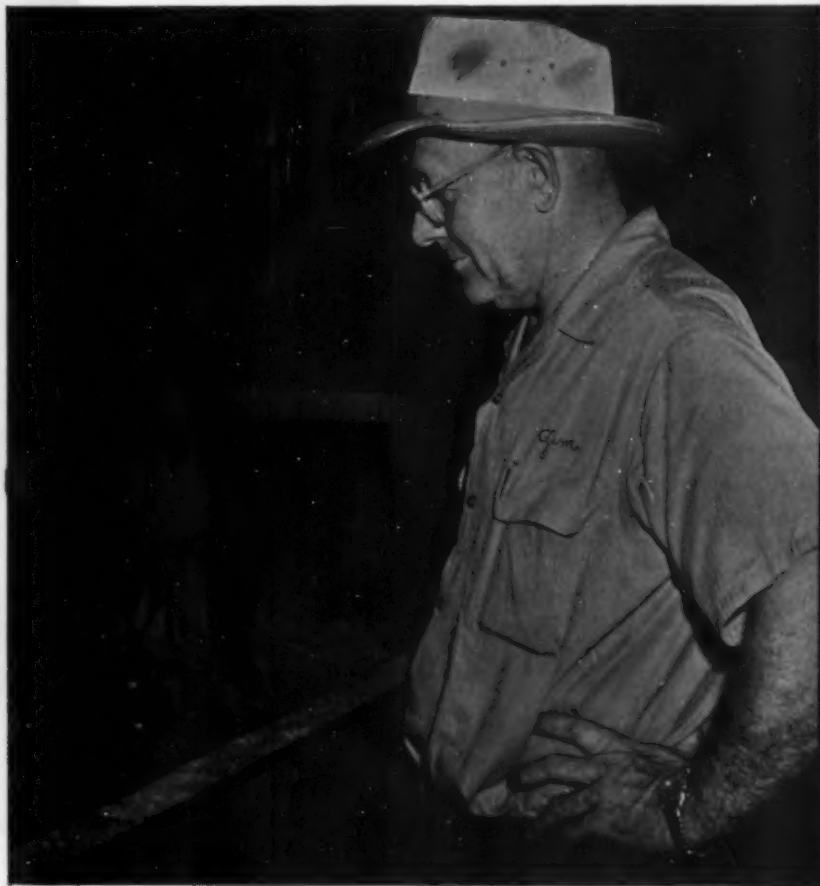
A Familiar Sign at Scenes of Progress

BUCYRUS-ERIE COMPANY • SOUTH MILWAUKEE, WISCONSIN

From the lab to the mill...

Whether you are developing the flow scheme for a new ore body or seeking ways to improve your present reagent combination, Cyanamid Field Engineers can give you very practical help.

They can bring you access to a vast reservoir of data on processes and reagent combinations proved most effective in mills all over the world. This operating data is supplemented by the work of Cyanamid Mineral Dressing Laboratory where basic research on better reagents and application methods goes forward unceasingly.



BEAUTY...OR BEAST?

We'd be the last to claim that Amsco® Dippers have any inherent glamour. That's why we induced the pert 115-pound* blond to pose in one of our 7-yd. dippers at a recent trade show. Got a lot of attention, too!

But we know that experienced operators *specify* these all-cast Manganese Steel Dippers because they're real "beasts"—when it comes to strength and ruggedness. By way of proof, glance at the other photo. It shows a 9-yd. Amsco Dipper hoisting a 60 to 70-ton* rock out of the St. Lawrence Seaway Channel. There's plenty more proof available, too—on the extra strength, digging ability and long life of Amsco Dippers. That's why they're the number one original-equipment choice on most leading makes of power shovels!

Consult your equipment manufacturer, or write us direct for new booklet on *Amsco Power Shovel Equipment*, giving detailed reasons why Amsco Dippers are real "beasts" when it comes to rugged digging work.

*Estimated

AMERICAN Brake Shoe COMPANY

AMSCO

American Manganese Steel Division • Chicago Heights, Ill.

OTHER PLANTS IN: DENVER, LOS ANGELES, NEW CASTLE, DEL., OAKLAND, CAL., ST. LOUIS; JOLIETTE, QUEBEC

Athey PR20 - Cat DW20 has 34-ton, 22.5 cu. yd. capacity. Four-wheel tractor gives speeds up to 32.1 MPH and good roadability for long hauls.

Athey PR15-Cat DW15 has 22-ton capacity and top speed of 37.2 MPH for high-speed hauling and dumping.

Athey PR21-Cat DW21 features 34-ton, 22.5 cu. yd. capacity, articulated design, top maneuverability, standard or heated body.

Athey PD20-Cat DW20 has 30-ton capacity, speeds up to 32.1 MPH, dumps "on-the-go," for non-stop cycles.

Engineered

for your specific hauling needs

There's an Athey-Cat hauling unit designed for your particular job — no matter what the conditions or the materials you handle. There are rear dumps for maneuverability and high speed, side dumps for on-the-go dumping, bottom dumps for high volume hauling. All are engineered for specific hauling conditions.

Having just the right hauling unit on your job makes a big difference in your profit. And Athey-Cat Trailer-Tractor units always give higher production, lower maintenance, and a bigger profit on every load.

Call in your Athey-Caterpillar Dealer for an analysis of your job and an expert equipment recommendation. Find out how just the right hauling unit can save you money. Athey Products Corporation, 5631 West 65th Street, Chicago 38, Illinois.

Athey

THE *Complete* TRAILER LINE... *by the Leader*

Western Uranium Output for First Half of 1957

Further information on the domestic uranium industry has been released by the AEC. The following statistics for the first six months of 1957 cover uranium concentrate production, ore production, stockpiles, ore reserves, bonus payments, and employment figures; they were compiled by the Grand Junction Operations Office.

Uranium Concentrate Production

During the first six months of 1957, 12 uranium processing mills were operating in the West, including one Government-owned mill at Monticello, Utah. Uranium concentrates received at the Grand Junction depot totaled as follows:

Month	Tons of U_3O_8
January	594
February	587
March	735
April	737
May	807
June	681
Total	4141

The following five new uranium processing mills are scheduled to go into production before the end of this year: Texas-Zinc Minerals Co. mill at Mexican Hat, Utah; Western Nuclear Corp. mill at Split Rock, Wyo.; Dawn Mining Co. mill at Ford, Wash.; Union Carbide Nuclear Co. mill at Rifle, Colo.; and Trace Elements Corp. mill at Maybell, Colo. Seven additional mills, including one pilot plant operation, are now under contract or construction.

Rate of Ore Processing

Uranium processing plants in the West treated ores at the following approximate daily rates during the first six months of 1957:

Month	Ore, Tpd
January	9000
February	9400
March	9300
April	9200
May	9600
June	9600

Domestic Ore Production

Uranium ore receipts at all domestic plants and Government purchase depots for the first half were received as follows:

Month	Dry Short Tons
January	269,000
February	262,000
March	296,000
April	281,000
May	297,000
June	301,000
Total	1,706,000

Ore Fed to Process

Mill feed was supplied in the following tonnage during the first six months of the year:

Month	Tons	Average U_3O_8, Pct
January	279,000	0.28
February	264,000	0.29
March	288,000	0.29
April	277,000	0.27
May	297,000	0.27
June	289,000	0.26
Total	1,694,000	0.28

Ore Stockpiles

Ore stockpiles in the western U. S. as of June 30, 1957, were as follows:

	Dry Tons	Average U_3O_8, Pct
Private firms	448,000	0.27
Government	1,498,000	0.28
Total	1,946,000	0.28

Ore Reserves

Measured, indicated, and inferred uranium ore reserves total an estimated 67 million tons with average uranium oxide content of 0.27 pct. These reserves include only uranium ores considered economically mineable and metallurgically amenable to treatment. Their distribution by states is approximately as follows:

State	Thousands of Tons	U_3O_8, Pct
New Mexico	47,700	0.26
Utah	5,700	0.36
Wyoming	5,300	0.24
Colorado	3,300	0.34
Arizona	1,800	0.29
Washington, Oregon, and Nevada	1,500	0.25
North and South Dakota	600	0.25
Texas, California, Montana, and Idaho	1,100	0.23
Total	67,000	0.27

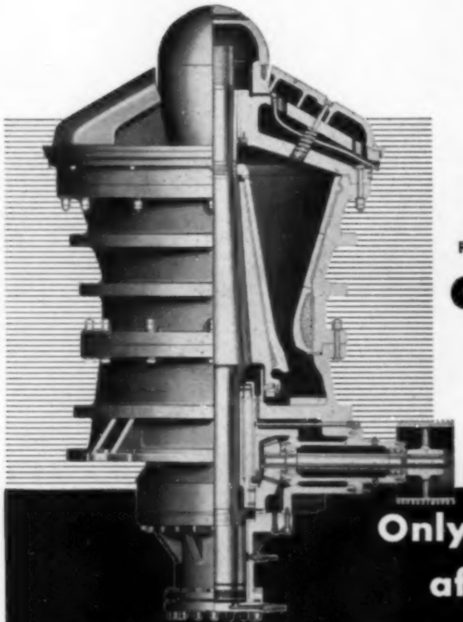
Initial Production Bonus Payments

Through June 30, 1957, a total of $10,581,062 had been paid on 832 properties as initial production bonus under the terms of Domestic Uranium Program Circular 6. Payments for the first six months of 1957 were as follows:

January	$383,490
February	165,489
March	187,115
April	211,678
May	338,573
June	213,974
Total	$1,500,319

Industry Employment

As of June 30, 1957, there were about 2800 persons employed in the uranium mills of the western states. It is estimated that some 5500 persons are employed directly in uranium mining in the West at this time.



Superior

Primary and Secondary

Crusher

Bulletin 07B7870

• *Retains initial setting* by compensating for wear on crushing surfaces with one-man, one-minute *Hydroset* mechanism.

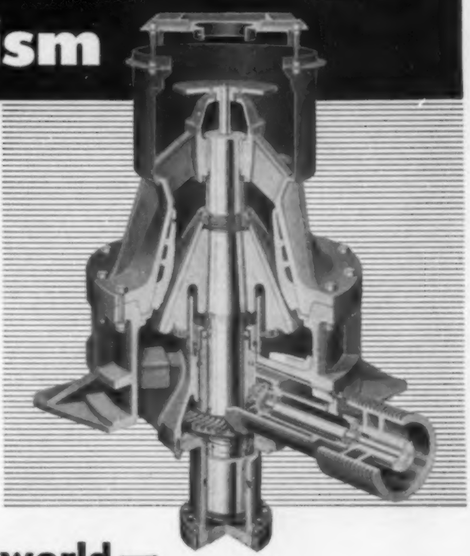
• *Entire circuit remains in balance.* Feed size to subsequent equipment remains uniform.

• *Aid in starting* is provided after crusher stops under load. *Hydroset* control reduces torque requirements. Clearing of the crushing chamber is facilitated.

• *High capacity* is the result of a unique crushing chamber design and *Hydroset* control of mainshaft position.

Only Allis-Chalmers gyratory crushers afford the many advantages of

Hydroset Mechanism



Hydrocone

Secondary and Tertiary

Crusher

Bulletin 07B7145C

• *Wide range of settings* at the flip of a switch is made possible by the *Hydroset* mechanism.

• *Chamber is cleared* — if crusher stops under load — by merely lowering the mainshaft hydraulically with the *Hydroset* mechanism.

• *Protection against tramp iron* is provided by *Hydroset* mechanism and Automatic Reset. After tramp iron is released, crusher returns to original setting. Safe — no spring-loaded chamber.

• *Compensation for wear,* as in the primary crusher, is a one-man, one-minute operation with *Hydroset* control.

Proved all over the world —

Limestone in China, iron ore in Africa, lead zinc in South America—gravel, dolomite, cement clinker, copper, perlite, trap rock — name the product, name the location, you'll find the *Hydroset* mechanism on the job proving its tremendous profit advantage. Hundreds of Allis-Chalmers gyratory crushers, processing the widest range of products, are equipped with *Hydroset* mechanism. Even the world's largest crusher, crushing hard taconite, features the convenience and economy of this remarkable design feature.

See your Allis-Chalmers representative or write Allis-Chalmers, Industrial Equipment Division, Milwaukee 1, Wisconsin.

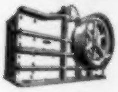
Superior, Hydrocone and Hydroset are Allis-Chalmers trademarks.

ALLIS-CHALMERS

A-C ALLIS-CHALMERS

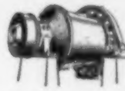
Hammermills

Vibrating Screens

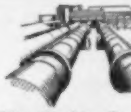


Jaw Crushers

Gyratory Crushers



Grinding Mills



Kilns, Coolers, Dryers

A-5322

NORDBERG MACHINERY used by leading TACONITE producers

To provide adequate iron ore reserves for today's increased steel production, the large-scale beneficiation of Taconite ore has become imperative.

Long used in the economical production of Taconite concentrates in the Arctic region of Norway, Nordberg Machinery is now serving the huge Taconite operations in northern Minnesota . . . where a new era in U.S. iron ore production is now underway.

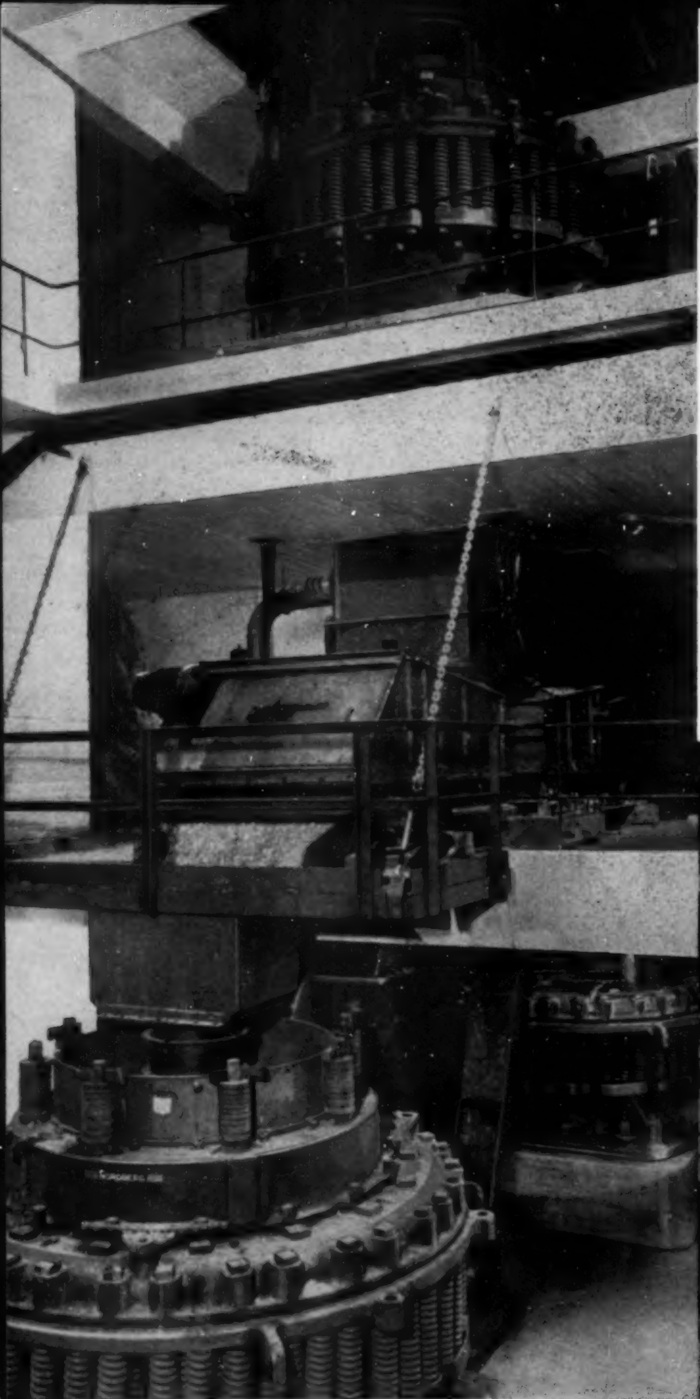
The selection of Nordberg Machinery for the processing of the extremely hard Taconite ore again follows the trend of the mining fraternity to rely on this heavy duty machinery for its proven ability to reduce ores and minerals at low per ton cost.

Nordberg Machinery now in use by leading Taconite producers includes Symons® Primary Gyratory Crushers, Symons Cone Crushers, Nordberg Rod and Ball Mills, and Symons Rod Deck Screens.

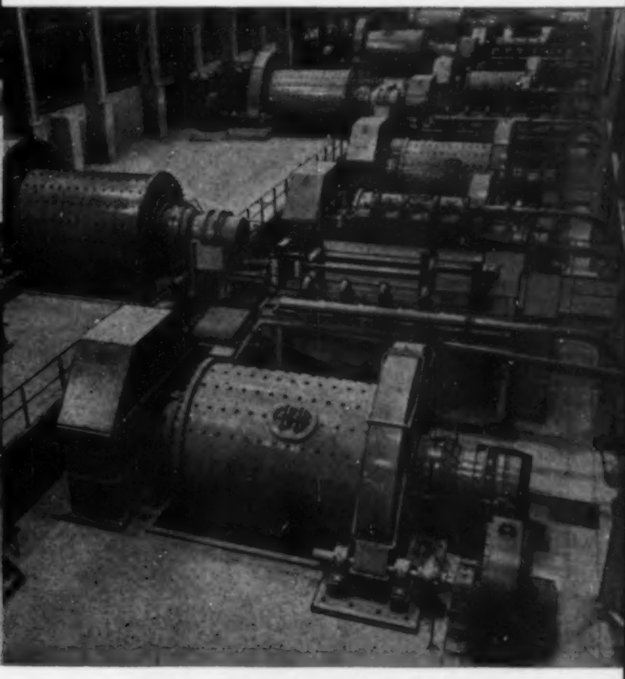
While *your* crushing and processing problems may not involve Taconite, you may be sure that Nordberg machinery is the logical choice for practically any other ore and mineral processing operation where big tonnages at low cost are required.

This Symons Primary Gyratory Crusher has a feed opening of 54″ and capacity of 2000 or more tons per hour of hard, abrasive Taconite ore. Symons Gyratories are built in sizes up to 72″ feed openings, with capacities of over 3500 tons per hour.

NORDBERG MFG. CO., MILWAUKEE,



Typical arrangement of Symons Cone Crushers and Symons Vibrating Rod Deck Screens installed for secondary and tertiary crushing, and screening of Taconite ore. Cone Crushers of this type are built in sizes up to 7′ diameter, with capacities to over 900 tons per hour.



Battery of four Nordberg 10½′ x 16′ Rod Mills and eight Nordberg 10½′ x 14′ Ball Mills installed in a large Taconite concentrating plant. Other Nordberg Grinding Mills are built in sizes from 6′ to 13′ diameter and up to 50′ in length in Rod, Ball, Pebble, Tube and Compartment types.

SYMONS . . . A REGISTERED NORDBERG TRADEMARK
KNOWN THROUGHOUT THE WORLD

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MACHINERY FOR PROCESSING ORES and INDUSTRIAL MINERALS

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M257

CANADA, according to a recent report, is putting some emphasis on the biogeochemical method of prospecting. The science, which depends on abnormal quantities of minerals in the stuff of various types of plant life to indicate underlying mineral deposits, is said to have been used successfully in locating concentrations of copper. According to L. W. Morley of the Canadian Geophysics Div., the method is well regarded by some prospectors in British Columbia.

Laboratory tests call for burning twigs or leaves in a spectrograph for comparison with the flame spectrum of the metal sought by the prospector. Matching lines suggest a more searching look at the indicated area. Because of plant affinity for only certain metals, the method is useless for finding others except in the case of possible associated deposits. Lead, zinc, and copper seekers should have the most success.

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CONVERTING the chemical energy of gases into usable electricity has been a long-nurtured laboratory hope. Now scientists at National Carbon Co. research laboratories have changed the aspiration to a practical reality.

Using hydrogen and oxygen as fuel, an electric cell just developed by the Union Carbide division employs chemically treated, hollow, porous carbon electrodes to introduce the gases to an electrolytic bath. The chemical reaction produces electric current.

"This new fuel cell," says K. Kordesch, who has been responsible for the research work, "is merely a sealed jar into which are fed hydrogen and oxygen through the special hollow electrodes. The electrochemical reaction of the gases at these electrodes produces an electric current, with only water as a byproduct. With the water disposed of by evaporation, the life of the fuel cell is theoretically unlimited. Cells have been operating here 8 hours a day, 5 days a week for the past year, with no signs of deterioration. This type of operation was purposely chosen for testing because the repeated starts and stops are much harder on the cell than would be continuous, around-the-clock operation."

Ability of the new fuel cell to operate at about atmospheric pressure has the advantage of eliminating heavy, costly pressure vessels. Increased output, however, can be obtained by increasing the pressure. For a given cell, higher outputs vary directly with pressure. Research to date indicates that optimum fuel cell design will result in one which will produce about 1 kw of power from a packaged unit of 1 cu ft in volume. Basically, the fuel cell is most desirable for high current, low voltage use.

Pure oxygen is needed for higher current densities, but the new cell can be operated with hydrogen and air for producing smaller amounts of power. This is convenient for small mobile units which can be operated simply on a tank of hydrogen and the surrounding air. Standard grades of industrial hydrogen may be used.

Inherent advantages of the fuel cell make it suitable for use in remote locations where conventional fuels of water power are not available.

One disadvantage is that the fuel cell is dependent on a practical and economical source of hydrogen, a gas that at present is quite expensive and requires bulky pressure vessels. Combining the unit with an industrial process might solve the problem. An inexpensive source of hydrogen is a must if the cell is to compete with conventional power systems.

An interesting possibility is the use of solar and nuclear energy to power the unit. Energy of the sun could be used to decompose water into hydrogen and oxygen, for later use in fuel cells. Nuclear energy is known to decompose water, and in present nuclear plants efforts have to be made to prevent this decomposition. Instead, nuclear energy might be used to produce directly the gases necessary for operating the fuel cell.

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PROJECTED mill construction to process the uraniferous lignites of North and South Dakota will be deferred "pending development of improved ore processing or a greatly increased demand for uranium oxide," according to Ohio Oil Co. and Arthur E. Pew. These interests which had proposed the operation told the AEC that their process studies showed it would not be economically feasible to follow through at the ceiling price of $10.50 per lb of uranium oxide contained in acceptable concentrates derived from lignite.

Since the mill was first proposed, reserves of richer minerals which could be processed more cheaply have been found. The AEC, however, offers the $10.50 per lb price to other prospective producers of uranium from lignite and currently is studying a preliminary proposal submitted by International Resources Corp. of Custer, S. D.

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ENGINEERS and scientists of the Geological Survey have gone the way of the push button and as a result we have better understanding of the way minerals are formed—thus providing clues to the discovery of new deposits.

An electronic digital-type computer system used by the Survey in recent months has produced new knowledge of the crystallographic structure of colemanite, meyerhofferite, and certain laboratory-made boron compounds. Stability of the minerals was thus predicted and applied in conjunction with petrologic studies to learn the cause for certain mineral sequences found in saline deposits of southern Cali-

fornia. By the use of computed data derived from X-ray studies it was possible to find out, for example, how certain ring-like groups of atoms in meyerhofferite become linked into chains in forming colemanite.

Current work goes even further. Researchers can now determine the precise locations of atoms in crystals, a procedure involving great numbers of complex computations.

The Survey's electronic brain consists of three interconnected units—an electronic computer, a magnetic tape memory or auxiliary storage unit, and an input-output system which prints, punches, and reads cards. The computer uses 10-digit numbers, computes 30,000 additions or subtractions a minute and in the same length of time can complete 7200 multiplications or 5100 divisions.

Use of the machine has extended the effectiveness of research by solving problems in a week or two that would ordinarily take years of concentrated mathematical effort. Voluminous information that might otherwise be lost through human failure can also be retrieved and applied.

The Survey expects later applications of the computer to include the analysis of more than half a million reports on geologic samples; coal resource computations; problems in ground water flow; flood-routing studies; geodetic positioning; and water flow duration studies of stream gaging records. Some 36 different projects have already been outlined for analysis.

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PETROLEUM is to a large extent a self-financing industry according to the views of at least two market analysts. Of the $24 billion of capital spent in the U. S. for petroleum facilities during the five years 1951 to 1955, $20 billion came from retained earnings and capital extinguishment charges, with only $4 billion provided primarily by the capital markets. This pattern of reinvestment of earning, report F. G. Coqueron and J. E. Pogue of Chase Manhattan Bank, is essential to support the present and anticipated growth of the petroleum industry.

Rehabilitation and expansion undertaken by the industry throughout the Free World after World War II increased gross investment in fixed assets by 174 pct, from $23 billion at the end of 1945 to $63 billion at the close of 1955. During the same period the daily rate of petroleum consumption expanded from 7.1 million bbl in 1946 to 14.4 million bbl in 1955, or more than 100 pct.

Capital expenditures after the war and up to 1955 amounted to $56 billion. Some $38 billion or about 68 pct went for properties and plants in the U. S., while $18 billion or 32 pct was spent for facilities in foreign areas. Total outlay was distributed as follows: 59 pct for production facilities, 13 pct for transportation, 16 pct for refining, and 12 pct for marketing.

More than $40 billion or 64 pct of the total investment employed at the end of 1955 was located in the U. S. The overall investment of $63 billion was divided among departments as follows: production 48 pct, transportation 17 pct, refining 20 pct, and marketing 15 pct.

The analysts emphasize a point: ". . .activities of the petroleum industry can be operated efficiently only through the agency of its integrated character, while essential competition can best be preserved through the medium of independent corporate entities."

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THE guided missile business, like most others, has very quickly developed a language all its own. The list of terms below supposedly results from the influence of former German scientists on the U. S. staff and was devised for use in technical literature. Personally, we prefer jumbos and jackhammers.

We are told *die echte* German script is not available—please pardon our wedding text.

NUCLEAR RESEARCH	Das Whizkidden Grupe.
PRELIMINARY DESIGN	Das Uppen-das-Klaudsen Grupe.
DESIGN	Das Raundscholder und Reddischeix Grupe.
PROJECT ENGINEER	Das Schwettenoudter.
MANAGEMENT	Das Ulxerenbalden Grupe.
ADMINISTRATION	Das Oudtgeschmardten Grupe.
GUIDED MISSILE	Das Skientifiker Geschtenwerkes Firenkrakker.
ROCKET ENGINE	Dar Firenschpitter mit Schmoken-und-Schnorten.
LIQUID ROCKET	Das Skwirten Jucenkind Firenschpitter.
CELESTIAL GUIDANCE	Das Schruballishe Schtargaxen Peepenglasser mit Kowputenratracen Schteerenwerke.
PRESET GUIDANCE	Das Senden Offen mit ein Pattenbacker und Finger Gekrossen Schteerenwerke.
WARHEAD	Das Laudenboomer.
NUCLEAR WARHEAD	Das Eargeschplitten Laudenboomer.
HYDROGEN DEVICE	Das Eargeschplitten Laudenboomer mit ein grosse Holengraund und Alles kaput.

"Thiokol" synthetic rubber, is an organic polysulfide elastomer. One of its many uses is in solid propellents for long range and high altitude missiles. In liquid form, "Thiokol" synthetic rubber mixed with an oxidizer, is poured into specially designed combustion chambers of rockets. It helps to give stability to the fuel charge and resistance to shock. It promotes uniform burning. When the rocket motor is ignited the mixture burns with great intensity and generates large volumes of gas to propel the rocket.

Solid propellents made with "Thiokol" synthetic rubber have proved their value in rockets over liquid propellents in many ways: they are less costly and easier to manufacture—simple and rugged construction makes handling and launching easier and safer—fuel tanks and complicated feed systems are eliminated.

"Thiokol" synthetic rubber is a product containing a high percentage of Sulphur—its name being derived from the Greek words for sulphur and glue. Here is another example of the continually broadening field in which Sulphur is an important and necessary element.

**A trade name of Thiokol Chemical Corporation.*

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811 Rusk Avenue, Houston 2, Texas

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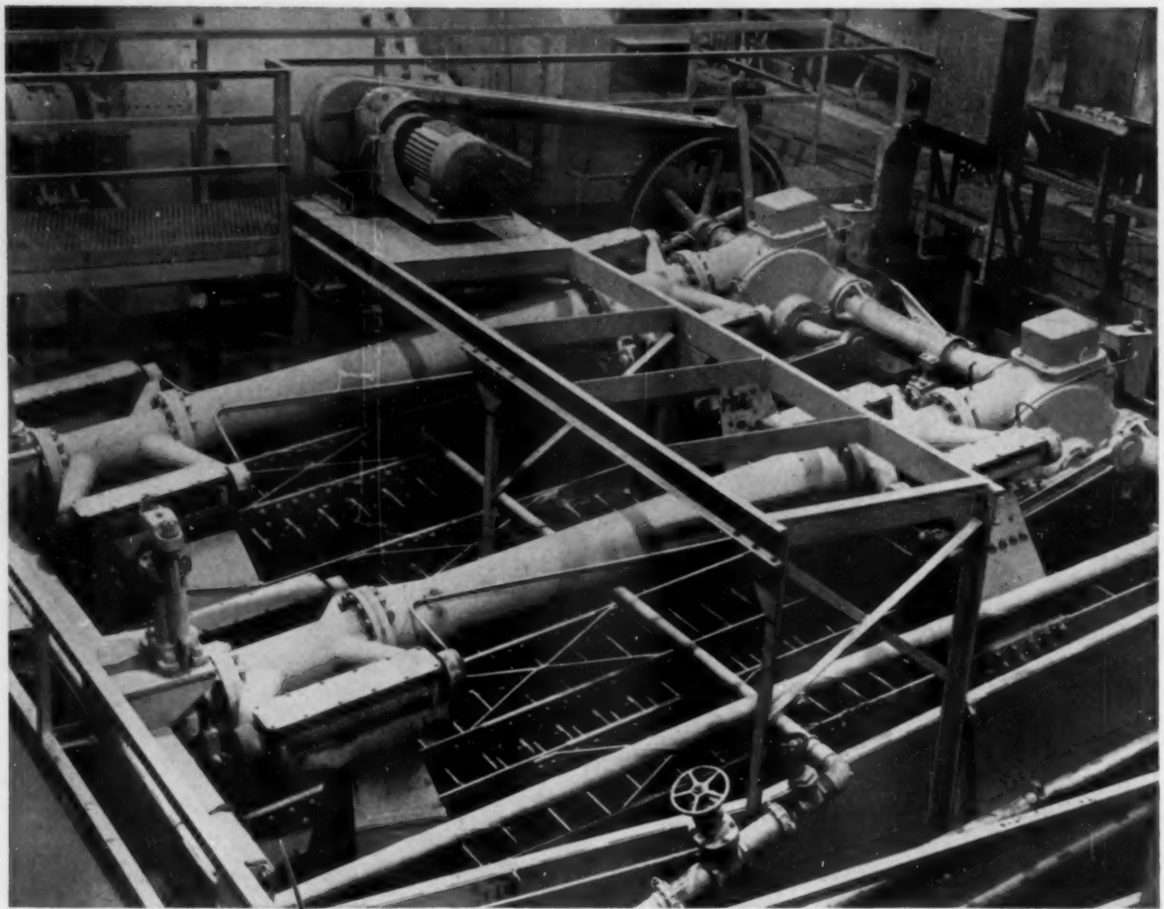
- Newgulf, Texas
- Moss Bluff, Texas
- Spindletop, Texas
- Worland, Wyoming

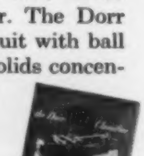


Idarado Mining Co. Classifies Complex Ore Body with **DORR** heavy duty Classifier

Rocky Mountain Mill Beneficates 1750 TPD of complex ore containing copper — lead — zinc — gold & silver

Situated high in the Rocky Mountains in Colorado almost 9000 feet above sea level, the Pandora Unit of the Idarado Mining Company has encountered one of the most complex ore bodies in the Western United States. Consisting mainly of copper, lead, and zinc sulphides, the ore also contains recoverable fractions of gold and silver.

In the Pandora flowsheet an 8′ x 12′ rod mill receives 1750 TPD of minute 1″ copper-lead-zinc-ore. The rod mill discharge is jigged to recover gold and silver which is then sent to amalgamators. The balance of the rod mill discharge, which is approximately 30% minus 65 mesh, overflows to the Dorr Quadruplex Classifier. The Dorr Classifier, which is 16′ wide by 28′ long, is in closed circuit with ball mills and produces a 65 mesh separation at an overflow solids concentration of 35%. Overflow product goes to a conventional flotation circuit for subsequent recovery of the copper, lead, and zinc values.



For more information on Dorr HX Classifiers, write for a copy of Bulletin No. 2281. Dorr-Oliver Incorporated, Stamford, Conn.

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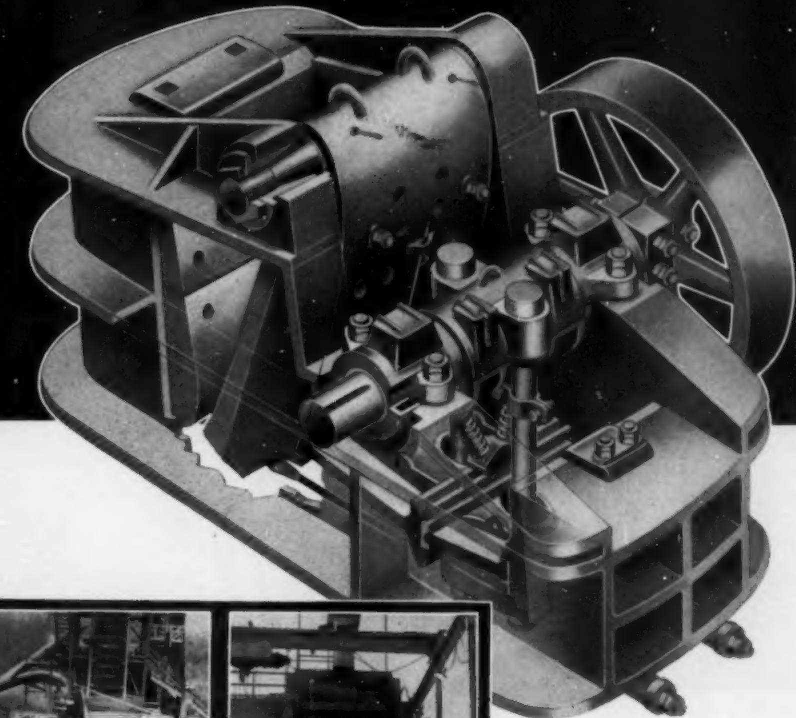
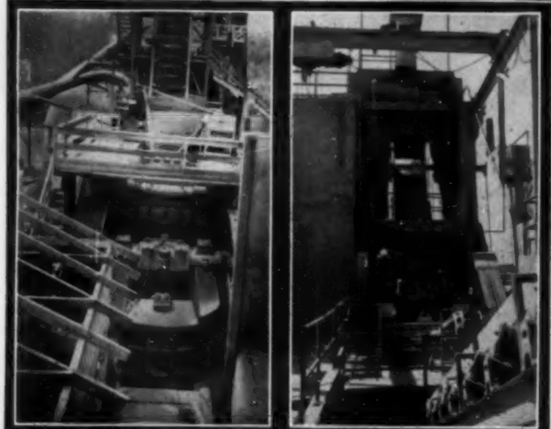


Type HB
Jaw Crusher



48" x 60" type HB Jaw Crusher in primary service, reducing quarry material to desired size.

Cutaway view of the type HB Jaw Crusher shows the advance design characteristics of Traylor-Made Jaw Crushers—notice the sturdy bulldog pitman. One of the many important features of this crusher is the Traylor curved jaw plate. This design employs the principal of the famous curved fittings which have proved so successful and satisfactory in Traylor crushing machinery. Write for Bulletin # 5105 for information.

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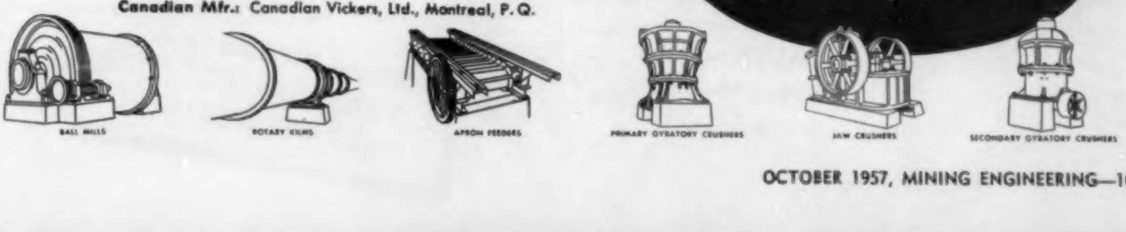


Trends in Professional Income of Engineers

An Engineers Joint Council Staff Report

EJC conducted surveys in 1953 and 1956 to determine the salary structure of the engineering profession. The results were published separately as *Professional Income of Engineers* in 1954 and 1957. The survey is the most comprehensive and representative study generally available on the subject. The EJC staff responsible for the statistical compilation and analysis of the data of the surveys have completed a comparison of the two studies. The summary results of the comparison are presented herein.

The 1956 sample was greatly expanded over that for 1953 (107,832 vs 69,061); however, the data of each survey are essentially comparable. The increase was due in part to a considerable expansion of engineering staffs of the participating organizations and in part to new participants in the 1956 survey. A review of the returns of these new participants has been made by the EJC staff and it was found that there were no material deviations in their data from that of the overall salary structure. The conclusion of the staff is that the surveys are comparable.

There are some variations in trends when each of the various industrial classifications of the survey are compared, but it would be most productive and meaningful, at this time, to analyze the structure of the entire industrial sample for both surveys. This means we shall consider the salary structure of all engineers covered in the various industrial and private activities (excluding thus, governmental and educational engineers). For ready reference the data were presented in Table VI, p. 14, of the 1953 study and Table VI, p. 15, of the 1956 survey. The various line groupings in the salary structure—median, quartiles, and deciles—follow, with minor variations, the same general trends; and for the sake of simplicity, the **median** line will serve as a basis of comparison. The relative change in salary will be reviewed in terms of percentage increase above the 1953 levels.

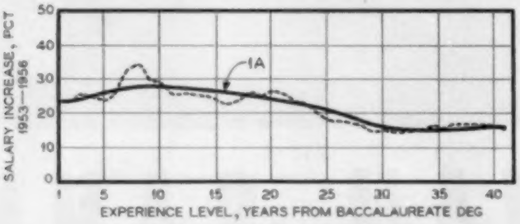
The first consideration will be the increment for each experience level or position—that is, the median salary for engineers with one year experience (referred to years from baccalaureate degee) in 1953 will be compared with the median salary for engineers with one-year experience in 1956. This would mean comparing what the 1952 graduate was earning in 1953 with what the 1955 graduate was earning in 1956. The same basis is used all along the line for all experience levels. The increases (as percentage) have been computed and plotted in **Graph 1.** It is shown that, for engineers with up to 22 years of experience, the salary rise was over 24 pct, reaching a maximum of 35 pct at eight years experience; this trend tapers off from the 22-year level (22 YFD—year from degree, or experience) and levels out for the older engineers at about 17 pct rise.

It should be noted here that the engineering profession is composed, in the majority, of a youthful group. Therefore, the period covered between 0 and 22 YFD (where the largest increments occurred) contains probably 75 pct of the engineering population. The curve labeled *1A* is an *average line* of increase which has been drawn to smooth out the fluctuations and indicate the overall trends.

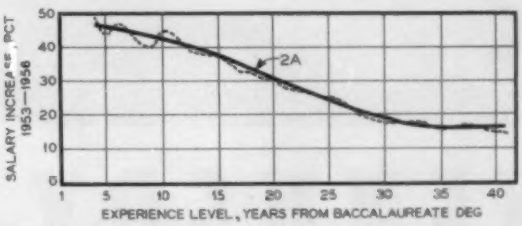
The next consideration is based on a comparison of the increments for each group of engineers—to trace the career salary progression. In this case the engineers who graduated in 1952 and had one year experience in 1953 are the same represented as having four years experience in 1956 and their relative salary movements are studied. The computed salary increments are plotted in **Graph 2.** It can be observed again that the larger increments occur in the areas of greatest population density, covering the same span as noted previously. Up to 20 YFD, the increment is above 30 pct, in fact, well above 40 pct for those with up to 13 years experience. It does again taper off at about the same level as for the position comparison.

For the same period (1953-1956), U. S. Dept. of Commerce figures indicate that earnings of all full-time wage and salary employes rose 12 pct and of full-time manufacturing wage earners, 13 pct. Bureau of Labor Statistics figures indicate that, for the same period, the Gross National Product increased about 13 pct, the Wholesale Price Index 4 pct, and Consumer Price Index 3 pct.

Conclusion: An impressive upward adjustment occurred in engineers' salaries between 1953 and 1956. To the best knowledge of the staff of EJC responsible for these studies, this rise is unparalleled in the recorded history of the engineering profession.



Graph 1. Salary increases as percentages have been computed and plotted in relation to position or experience. The base year is 1953 and the increase curve is based on the year 1956.



Graph 2. Salary increase was plotted as it related to individuals. Again the base year was 1953 and the increase curve plotted was based on the year 1956.

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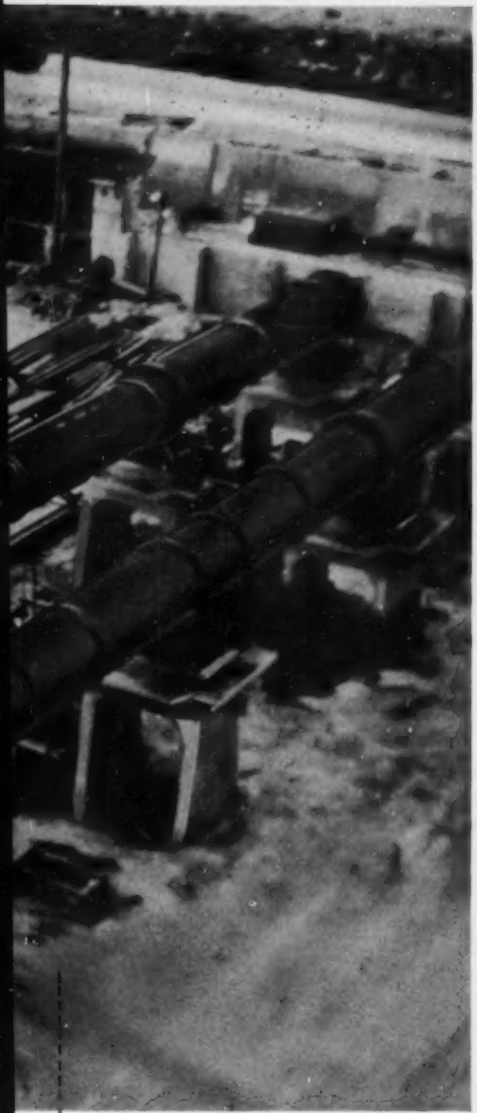
Plant . . . Capacity 5000 barrels per day
COMPANY, Island of Java, Indonesia



Aerial view of Gresik Cement Company Plant, Island of Java, Indonesia

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- Classifiers
- Pneumatic Transport Systems
- Dust Collectors
- Complete Cement Plants
- Complete Lime Plants
- Waste Heat Boilers
- Pulverized Coal Firing Systems

NEW CEMENT PLANT OPENS IN INDONESIA

Special to The New York Times.

JAKARTA, Indonesia, Aug. 7—Asia's most modern cement plant, an impressive example of Indonesian-United States cooperation, was formally dedicated today by President Sukarno in East Java.

The ceremony in the newly constructed industrial compound at Gresik, near Surabaya, represented an important event in the history of underdeveloped Indonesia, now struggling through her worst economic crisis. The cement plant, sprawling over several acres is expected to make a major contribution in helping strengthen the nation's economy and spur new industries.

From Gresik cement will be transported to scattered areas of the archipelago. Today's cement at Gresik will be tomorrow's new schools, houses, dams, hospitals, factories and bridges, all of which are badly needed by Indonesia's 82,000,000 people.

In a dedication speech, as reported here, President Sukarno thanked the United States for financial and technical cooperation in building the Gresik works.

Cement experts said the plant had a capacity of 250,000 tons of cement a year, about 75,000 tons more than total present production. They said the plant was expected to save Indonesia about $7,000 a day in foreign exchange.

Reprinted from
The New York Times, August 8, 1957

Send for Bulletins describing Kennedy Machinery and Equipment

Mining Coking Coal by Mechanized Methods

by John Peperakis and James Quigley

IN 1950 Kaiser Steel Corp. acquired control of the Utah Fuel Co., a pioneer Utah coal concern owning large reserves of high volatile coking coal near Sunnyside, Utah, and large reserves of coal elsewhere in Utah and Colorado. Kaiser Steel later disposed of the noncoking quality reserves included in the property and over the past six years has developed a group of three fully modernized mines and a central preparation plant at Sunnyside. This plant is geared to furnish 1.6 million tons per year of low ash coking coal to the byproduct coke ovens at its steel plant at Fontana, Calif., and to its beehive oven coking plant at Sunnyside.

General Problems: Topographic and physical conditions under which Utah coals are found differ from those in most bituminous coal districts of the eastern and central states. Utah coal seams lie under deeper cover, generally at steeper pitches; considerable distances are burned along the outcrop and back from the surface of the outcrop. Usually the coal seams lie high up on escarpments, accessible only through deep narrow canyons or long rock tunnels and tramways. The topography at Sunnyside is particularly rough, with great changes in cover over the coal in relatively short horizontal distances. This is due to the cliff-making sandstones of the cretaceous Mesa Verde group, the coal-bearing measures being in the Blackhawk formation. The thick massive Castle Gate sandstone of the Prince River formation overlies the coal-bearing rocks and is suspected to be the chief cause of roof control problems.

The Kaiser Steel property extends for nine and a half miles along the face of the Book Cliffs. The tract of coal land currently being mined extends six and a half miles along the outcrop. The coal occurs in two mineable seams, separated by as much as 45 ft of shale in the central part of the tract and only a foot or so in the southeastern part of the property. Where the upper seam has been mined it varies from 3½ to 5½ ft thick and in some areas is so split up it is not workable. The lower seam ranges from 5½ to 13 ft thick. Where the two seams come together in the southern part of the tract mining heights are often 20 ft, owing to the necessity of mining with the coal the split between the two seams as well as cap rock above the upper seam.

The Book Cliffs lie on the northeastern flank of the San Rafael anticline and on the southern flank of the Uinta Basin syncline. The coal seams dip in under the cliffs to the northeast at 6 to 22 pct. Faults roughly parallel to the strike with displacements up to 33 ft are common. Some faults also extend down the dip but are not as common.

Former Operations: The Sunnyside coal deposits were opened up by the Utah Fuel Co. in the late 1890's, the first production being recorded in 1900. The mines were worked intensively for the coals coked in beehive ovens at the property. During its peak period of production between 1910 and 1920 Sunnyside was one of the great coal producing camps in the West, but as demand for beehive coke diminished Sunnyside dwindled from the largest community in Carbon County to almost a ghost town in the 1930's, production being largely confined to a small amount of beehive coke to lead smelters. In 1942 when Kaiser Steel started construction of the steel plant at Fontana, Calif., part of the Sunnyside prop-

J. PEPERAKIS is Manager of Sunnyside Coal Mines, Kaiser Steel Corp., Sunnyside, Utah. J. QUIGLEY is Vice President of Centennial Development Co., Eureka, Utah.

Mine rehabilitation required extensive advances through old caved slopes since substantial reserves remained within areas cut up by miles of old rooms and entries. More than a million rock bolts have been installed since 1949, and some of the thousands of war surplus landing mats used to support soft shale may be seen on side walls of this entry.

erty was leased from Utah Fuel Co. This lease covered the southern part of the property, and the old No. 2 mine was reopened to establish a source of coking coal for the byproduct coke ovens then under construction at Fontana.

In 1950 Kaiser Steel acquired the entire Utah Fuel property and proceeded immediately to develop the mines into a fully modernized source for all its western coal needs. Low volatile midwestern coal is still purchased to blend with the Sunnyside coal to make a metallurgical coke that will stand up in a blast furnace. Recently Kaiser Steel purchased a 530,000-acre reserve of coking coal near Raton, N. M. Although only small amounts of coal are drawn from this source at present, it can be called upon to supply coking coal for any foreseeable demands by the western steel industry.

Problems Related to Former Operations: At the time Kaiser Steel took over the Sunnyside properties 21 million tons of coal had been mined from the upper seam, the lower seam, and in places the two seams together. These worked out and partially worked out areas covered 4 sq miles extending 5 miles along the strike and a maximum of 1½ miles down the dip of the seams. The better areas of clean coal and areas of good mining conditions under light cover had been mined out. Most of the reserves remaining were on the fringes of the worked out areas and under deeper cover and very difficult roof conditions. Substantial reserves also remained within areas cut up by miles of old rooms and entries. To gain access to these very large remaining reserves it has been necessary to reconstruct some 22 miles of old slopes and entries at great cost, a program now in its sixth year and nearing completion. It is estimated that ten more years of mining will be required to work clear of some of the deeper old workings.

The surface facilities required complete rebuilding and extensive relocation. A new central warehouse, a large maintenance shop, and a large hoist are currently under construction.

Shortly after the properties were acquired a blast furnace at Fontana was constructed to supplement the two already in service. This boosted by 50 pct the plans to build up production to serve the blast furnaces.

Early planning centered on: 1) development of haulage and ventilating facilities to support high production face mining and 2) reconstruction of the existing coal preparation plant to handle increased tonnage and higher amounts of waste in the raw coal without increasing the labor force. Then attention was turned to full mechanization at the face and gradual improvement of mining methods and operating machinery.

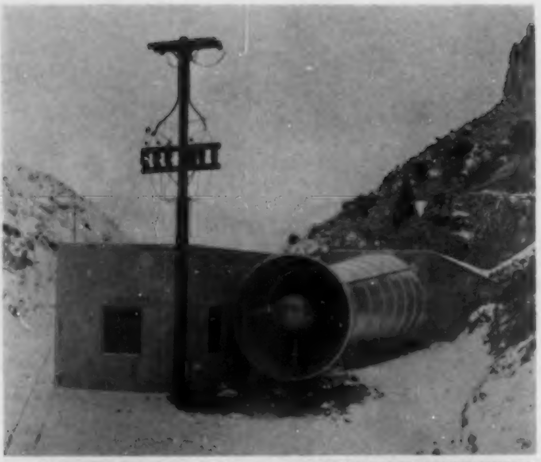
Yieldable supports are used in this section of the No. 1 mine main hoisting slope. Entry at this property is through an 1800-ft rock tunnel to the lower seam, a mile along the strike, and then via a rope slope down dip. Primary haulage and rope slopes get 80-lb rail with welded joints on 40-in. gage. Secondary haulage is on 60-lb steel.

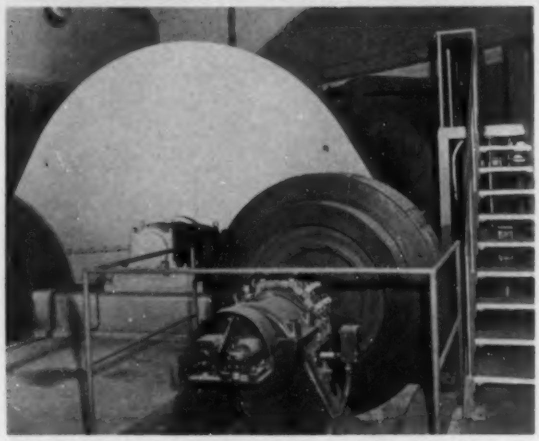
General Operating Plans: Three mines were laid out with separate ventilating and haulage systems. The haulage systems all converge on the automatic dumping facilities at the head of the preparation plant. No. 1 mine extends northwest through an 1800-ft rock tunnel to the lower seam, then 1 mile along the strike to a rope slope down the dip served by a 1000-hp hoist. A new area is being developed beyond this slope to the raise above an extension of the strike haulageway. No. 2 mine, southeast of the preparation plant, is served by a haulageway along the strike through 9000 ft of mined out areas to a rope slope down the dip served by a 600-hp hoist. No. 3 mine lies directly down the dip from the washing plant on a rope slope that will be served by an 800-hp hoist now being installed to replace a smaller unit that has reached its capacity.

Haulage: Since modern coal mining depends on efficient movement of mined coal from working faces to the surface, one of the first steps in the rehabilitation program was to construct a haulage system on the most favorable grade and alignment possible in each mine. This meant penetrating caved workings, regrading by rock excavation, and resupporting with roof bolting and with concrete, which is used in areas of excessive pressure. Eighty-pound steel with welded joints on 40-in. gage was laid on the 8 miles of primary haulage and rope slopes in the three mines. All secondary haulage is on 60-lb steel. On the main line 20-ton locomotives haul 50 car trips, each car holding 5.75 tons of raw coal. Speeds range up to 15 mph. All main haulage locomotives are in direct contact with a central dispatcher's office by trolley wire speaker system.

Entry haulage off the rope slope is secondary to the main haulage, since rail and ties are salvaged for further use as an entry is worked out. Entries are turned off on each side of the rope slopes on 350 to 400-ft centers and advanced outward to the mine boundaries. When an entry is developed to its full length the driving of rooms and extraction of pillars commences on the retreat. Trackage loading facilities and equipment are gradually moved backward as mining retreats to the barrier pillars protecting the slopes. The shortest room entries are 3000 ft, the longest 8000 ft, and the average 4500 ft. A two-heading entry driven by a continuous miner advances about 500 ft per month. Under favorable conditions where roof support is not a problem entries have advanced up to 1000 ft per month.

Ventilation: Modern methods mean rapid face advance. The ventilation problem is twofold: first, permanent air courses must be designed to integrate into an expanded future system, and secondly, new





TOP: One of two 8-ft Aerodynefans that deliver 380,000 cfm to the No. 1 mine. BELOW: The 1000-hp Nordberg hoist serving the main haulage slope.

construction of airshafts and airways must proceed ahead of demand to insure uninterrupted production.

No. 1, the most gaseous mine, making about 1.6 million cu ft of methane per 24 hr, is equipped with two 8M-84 Jeffrey Aerodyne fans each powered by a 150-hp motor. One of the fans is also equipped with a standby diesel drive that cuts in automatically in the event of a power failure. The two fans force about 380,000 cfm into the mine through a system of intakes parallel to the main hoisting slope. Air is returned through six airways converging to three large openings at the outcrop.

No. 2 mine is ventilated by a Joy model M-96 fan equipped with a 200-hp motor. A 200-hp motor used as a standby can be connected to the drive in a matter of minutes. A system very like that in No. 1 mine forces 350,000 cfm into the mine.

No. 3 mine is ventilated by a 150-hp drive fan exhausting 210,000 cfm. This is a very old installation, now being replaced by a large Joy model M-96 fan situated nearby at the top of a 16-ft diam airshaft 115 ft deep to the coal seam. The system of coursing air through the mine is similar to that used in No. 1 and No. 2 mines.

The presence of large old worked out sections and the heavy cover add to the ventilation problem at Sunnyside. Aircourses and haulageways passing through the old workings or adjacent to them must be sealed off from these workings to prevent dilution of the fresh air with black damp. Stoppings and seals require special construction to provide airtightness under great pressure.

A sizable force constantly maintains the aircourses. This has paid off in more efficient production.

Mining Methods: Because roof conditions at Sunnyside are extremely poor, mining operations are very difficult. The following table shows the percentage of production coming from various depths:

Depth, Ft	Percent	Tons Per Day
More than 2000	35	2450
1500 to 2000	11	770
1000 to 1500	15	1050
500 to 1000	39	2730

Although these depths are moderate compared to those reached in metal mining they are considered very deep in the American bituminous coal industry. The heavy cover and pitch of the seams induce such pressure in the abutment areas ahead of mining that it is almost impossible to hold the fragile roof in place by any type of support. In some of the worst sections heavy concentrations of roof bolts combined with very heavy timbering have failed to hold the top in place long enough for mining to be completed. When caves occur either forepoling or spiling with discarded drill steel is used to get through the caved mass of rock.

Since roof bolting was started in 1949, 1.265 million bolts have been installed and 135 miles of workings have been roof bolted. Current use runs from 20,000 to 22,000 bolts per month. Bolt lengths, varying from 4 to 10 ft, are usually 6 to 8 ft. War surplus pierced plank from wartime landing fields is used extensively to provide more bearing area for bolting to prevent the soft shale from falling out between bolts. Two thousand landing mats are installed monthly. Bolt spacing is 4 ft or less if required.

It has been found that areas of costly mining with difficult roof conditions cannot be bypassed for immediate production gains. Every effort has been made to insure recovery as complete as possible now and in the future. The Sunnyside reserves are considered an important asset worthy of substantial expenditures to insure low cost production and to insure maximum recovery as far as natural conditions permit.

Poor roof conditions greatly reduce flexibility in mining. It is not possible, as in some mines, to favor maximum loadability by devising mining systems with many working faces. Years of experience with modifications of the room and pillar system at Sunnyside have proved that maximum production is obtained with a minimum number of working faces. This reduces the system used to the room and pillar method in one of its most basic forms. Skill of the highest order is necessary in planning, supervision, and workmanship.

After the room entries have been driven to the limits of the coal block the operation of driving rooms and extracting pillars begins. In sections where workings are under heavy cover and under poor top the practice is to drive a single room 22 ft wide up the pitch through the block to the worked out area above. When the rooms break through, extraction of the pillar begins downward, usually fol-

Workings are deep by bituminous coal production standards (see page 1088), and heavy cover and pitch of the seams induce high pressure in abutment areas ahead of mining. In the worst sections heavy timbering plus rock bolting have failed to hold long enough to complete extraction, and forepoling or spiling is used to get through the caved mass or rock.

lowed closely by roof caving. Occasionally large areas stand for a week or so before caving, even though extraction has been very complete.

In the few areas where roof conditions permit, several rooms are advanced, pillar extraction following. Room centers depend on thickness of seams, roof conditions, and type of mining equipment used. These centers range from 35 to 50 ft, averaging 40 ft. Minimum room centers are favored because the small pillar next to the cave crushes as the room is advanced, so that there is less likelihood of bounces or mountain bumps.

In areas where the seam is more than 10 ft thick entries and rooms are advanced in the upper part of the seam, leaving bottom coal that is recovered at the time pillars are extracted. Known as *top benching,* this system was pioneered at the Sunnyside mines and is now being used successfully in several other large operations in the state. Formerly, in high coal, entries and rooms were advanced in the lower part of the seam, leaving several feet of top coal for the immediate roof. Under heavy cover and abutment pressures induced by pillar mining, the top caved or had to be taken down and supported with bolts and long timber. This resulted in high and dangerous ribs. Bolting and timbering were slow operations because of the staging necessary to provide a platform for installing roof supports. Roof support was not very effective because the top rock overlying the coal had also broken up. The long timber served only for warning purposes and did not provide much support.

The change to the top bench system resulted in more moderate working heights. Installation of bolts was accomplished much faster—the top rock was bolted as fast as it was exposed by the advancing coal face. Eliminating the high ribs provided more security to personnel, who are now exposed to high ribs only in the immediate face area when bottom coal is taken up. In this area roof and rib support previously installed on the advance provides adequate protection. One of the important benefits resulting from the introduction of this system has been to lessen considerably the danger to personnel from mountain bumps. It has been found that where this system of mining has been properly applied in virgin territory the possibility of bumps has been reduced markedly.

Ten continuous miners are now in operation. These are well suited to conditions at Sunnyside because they require a minimum number of places to provide good production efficiently. With a seven-man crew peaks of more than 500 tons per shift have been reached, but the average is about 250 tons per shift, or 36 tons per man.

Conventional mining units are used in areas where rock splits prevent the use of continuous miners. Seven conventional units are in operation.

Despite difficult mining conditions a performance of 10 to 11 tons per man total payroll is average. This compares favorably with any other large coal mining operation in the state, commercial or captive.

Coal Preparation Plant: All coal mined is washed in a centrally located plant near the portal of No. 3 mine. With two 7¼-hr shifts, input of raw coal to the plant averages close to 9000 tpd—7000 tons of clean coal and 2000 tons of reject material sent to the waste disposal dump.

Two rotary dumps handle the coal cars from the three mines at 950 tph, and a separate rotary dump emptying into a rock bin handles mine cars loaded with rock. The mine-run coal is reduced to 6x0 size in a large 27x12 rotary breaker, and the product from the breaker is stored in a 4000-ton blending and surge bin. By means of eight vibrating feeders, 650 tph are taken from the bins on a 48-in. belt conveyor system to two Baum-type jigs. Primary cleaning in the two jigs is followed by cleaning of the middling product in a smaller Baum-type jig. Fine coal is cleaned in a flotation unit. Quality control is supervised by a coal preparation engineer, assisted by a sampler and chemist.

A 40,000-ton stockpile facility is provided to assure uninterrupted operation of the washing plant and mines during periods of railroad car shortages.

Silver-Lead-Zinc Mines at Namiquipa, Chihuahua, Mexico

MAP SHOWING SITUATION OF NAMIQUIPA CHIHUAHUA, MEXICO

by G. H. Shefelbine

DISCOVERED during the time of the Spaniards, the Namiquipa silver-lead-zinc mines in Chihuahua, Mexico, remained idle until 1916, when mining claims were first denounced. About 1922 a Japanese company, the Santo Domingo Mining Co., was formed to mine the oxide silver ore. The company built a 75-tpd cyanide plant and mined in the area for a number of years. Because of an error in surveying in one of the company's mines situated in another district, mining was carried over into a claim owned by General Marcel Caraveo, at that time governor of Chihuahua. Santo Domingo Mining Co. lost the resulting lawsuit and consequently had no capital to continue operating at Namiquipa.

Later, Ray Brown and Eugenio Caballero formed a company to operate the mine, shipping crude ore to the American Smelting and Refining Co. smelter at Avalos, Chihuahua. The cyanide plant was changed to flotation with a capacity of 75 tons per 24 hr. In 1936 the property was again closed owing to labor trouble, a drop in the price of silver, and increased water in the mine.

In all these operations, mining was confined to the rich oxide-silver zone, as the presence of zinc presented problems in mining the sulfide ore. H. C. Dudley and W. N. Fink had long realized the existence of large bodies of sulfide ore below the relatively shallow workings of former operators; they also inferred that mineralized areas existed to the south of those previousely explored. During 1946 and 1947 they directed a vigorous exploration campaign by diamond drilling both under the known ore zones and south of them. A new large orebody was found in virgin ground, and extensive lenses of sulfide ore were probed below the old workings. A modern milling plant capable of treating 500 tpd was started in March 1948, and actual milling began in September of the same year.

The nearest rail terminal and shipping point is Tejolocachic, Chihuahua, 43 miles southwest, to which a graded, all-weather road was built. The surrounding country is not desert as might be expected—the plains are dry-farmed for oats, corn, and beans, and the hills are good range for cattle.

Properties: The Cia. Minera Venturosa, S. A., formed by H. C. Dudley of Duluth, took leases with options to purchase the diversely owned mining claims located over the principal Namiquipa veins. The area was enlarged by new denouncements, and at the end of the exploration program in 1947 the mineral rights on 850 hectares (2100 acres) were owned or leased. In 1949 options were exercised to purchase the leased mineral claims, with the exception of the Princesa claim comprising 20 hectares (50 acres). By negotiations started in 1947, Cia. Minera Venturosa finally obtained surface rights to some 265 hectares (655 acres), including rights to the mill and surface plants.

Geology: The mines are at an elevation of about 6100 ft on the west flank of a prominent hill in the center of the wide Santa Maria River valley. To the west, about 12 miles across the rolling valley,

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the Babicora Mts. rise to a height of more than 8000 ft. To the east, at an equal distance, rises the south flank of the Sierra de las Tunas. South and east of the mine relatively recent andesite and rhyolite flows form low, rugged escarpments with wide valleys between.

The ore was found in steeply dipping quartz veins within andesite lava flows of Tertiary Age. The host rock is a hard, gray andesite flow of considerable thickness. The flow has been gently arched, and veins were usually found along the west limb of the arch. The particular andesite bed that served as host rock for the ore deposition is markedly different from the other andesite in the region in that it is much altered, having an increased silica content and containing pyrite in the form of fine crystals, well disseminated throughout the bed. On the surface the flow weathers to reddish gray or pink, tinged with violet that is probably due to the increased pyrite in and around the veins and to the large quantity of fluorite that occurs within the vein filling and adjacent to it. Erosion of the andesite bed has been greater over the veins than in the more siliceous part of the flow along the axis of the arch. This is also probably due to the greater abundance of pyrite there, which has caused more active chemical action during the weathering process.

The favorable andesite is overlain by more recent andesites and rhyolites and at the mine dips about 20° SW. At the south end of the mined zone the dip increases sharply to about 60°. There the bed is covered by a soft, clay-like andesite tuff or ash, often of a marked red color. The tuff bed is almost completely impervious to water and is at least 500 ft thick at the southwest end of the property.

The six principal veins of the area all strike about N 20° E at the north end of the property, but at the south end they fan out so that the easternmost veins have a strike of N 20° W. Dips near surface are fairly constant and vary from vertical 70°; when departing from the vertical, the most westerly vein dips to the east, and the veins east of that dip to the west. Usually the dips of the veins steepen with depth, until at 250 meters (about 800 ft) below the surface they are all nearly vertical.

Conversions

1 meter = 3.281 ft
1 liter = 0.264 gal
1 hectare = 2.471 acres
1 metric ton = 1.102 short tons

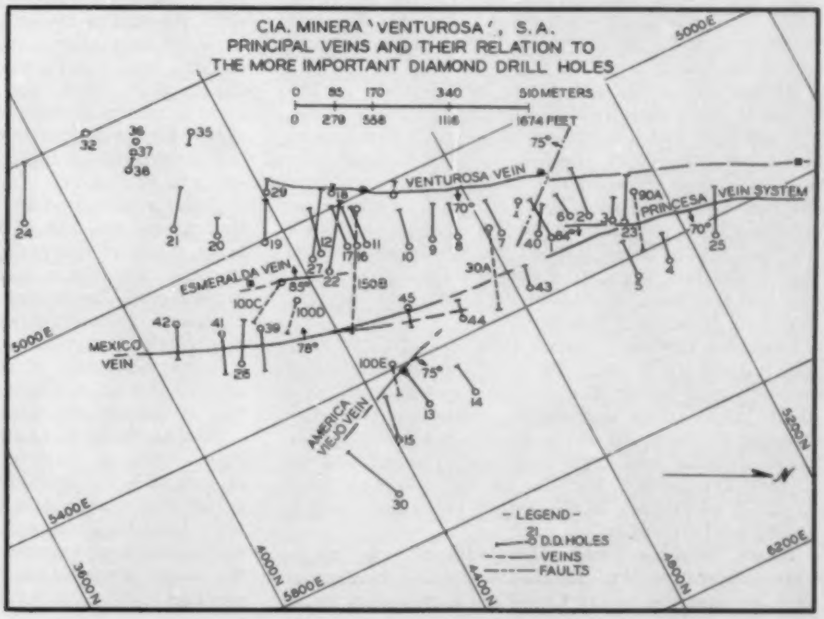
Vein fillings are essentially quartz replacements of breccia-filled fissures in the andesite. The largest vein, the Venturosa, shows evidence of several ages of mineralization, each with its distinct type of mineral. The chief vein constituents in the sulfide or oxidized zone are quartz, sphalerite, galena, pyrite, fluorite, and chalcopyrite with silver minerals and minor gold. Chalcedony is also present in many places.

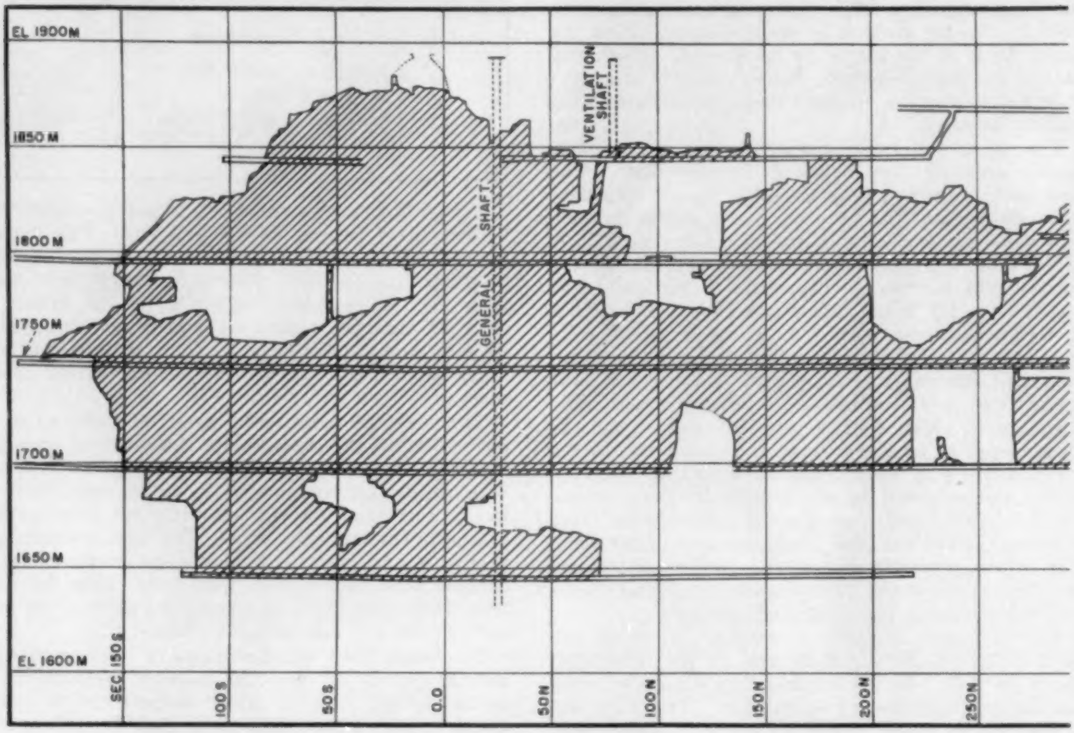
The veins are oxidized to about 100 meters (about 325 ft) below the surface. In the oxidized zone the lead and zinc minerals have been leached and the vugs resulting remain open or contain minor limonite, as a residual constituent resulting from oxidation of pyrite and sphalerite. The lead minerals are cerrusite and, rarely, anglesite or wulfenite. Increased quantities of barite are found near the surface where the silver minerals are native silver and cerargyrite.

The veins show distinct zones of silver and lead enrichment at or near the base of oxidation. A second broad zone of silver enrichment is found higher up at about 50 meters (165 ft) below the surface. In general the transition between oxides and sulfides takes place within several feet; however, the base of oxidation is very irregular, many tongues of oxide extending tens of feet into the sulfide zone. As would be expected, the richer, more open vein areas generally show greater depth of oxidation.

Outcrops of the veins are obscure and no gossans as such are evident. The large halo of discolored rock surrounding the area where the veins occur is a diagnostic feature of major consequence and one found in many western Sierra Madre camps that have yielded important ores.

Important ore reserves were found with the first 28 diamond drillholes put down from the surface. About 30,000 ft of surface and underground hole was drilled in the exploration program that continued throughout the life of the mine.





At the surface and in the completely oxidized zone of the veins the color of the residual limonite and the nature of the vugs or box works left by leaching give a clue to the amount of lead and zinc in the original veins before alteration. In general, limonite resulting from oxidation of galena is orange, whereas that resulting from sphalerite is more often tan or brown. The limonitic oxidation product of pyrite is nearly always brick red, but occasionally bright red. The term *limonite* is used to designate undistinguished iron oxides.

Several normal faults cut the veins at nearly right angles, but their displacement is not great. In these faults the south walls have invariably dropped relative to those on the north. Dikes are small and few in number.

At the south the ore in the vein system is believed to have been localized by the extensive andesite tuff bed, which is almost impervious to solutions and is not competent, mechanically, to support vein openings; thus the evidence supports the view that the tuff bed acted as a solution trap. Veins have nowhere been traced into the tuff zone more than a few feet before being lost. At the north end the ore zone is shallow, suggesting that the favorable andesite bed has been eroded to its base there, as disclosed in diamond drillholes.

Diamond Drilling: During 1946 and 1947, 28 surface holes were drilled. Important sulfide orebodies were proved below the original mined area, and much larger oxide and sulfide orebodies were discovered to the south of the former workings. Diamond drilling was continued underground and on surface throughout the life of the mine, a total 23,413 ft of drilling in 46 surface holes and 6548 ft in 26 underground holes.

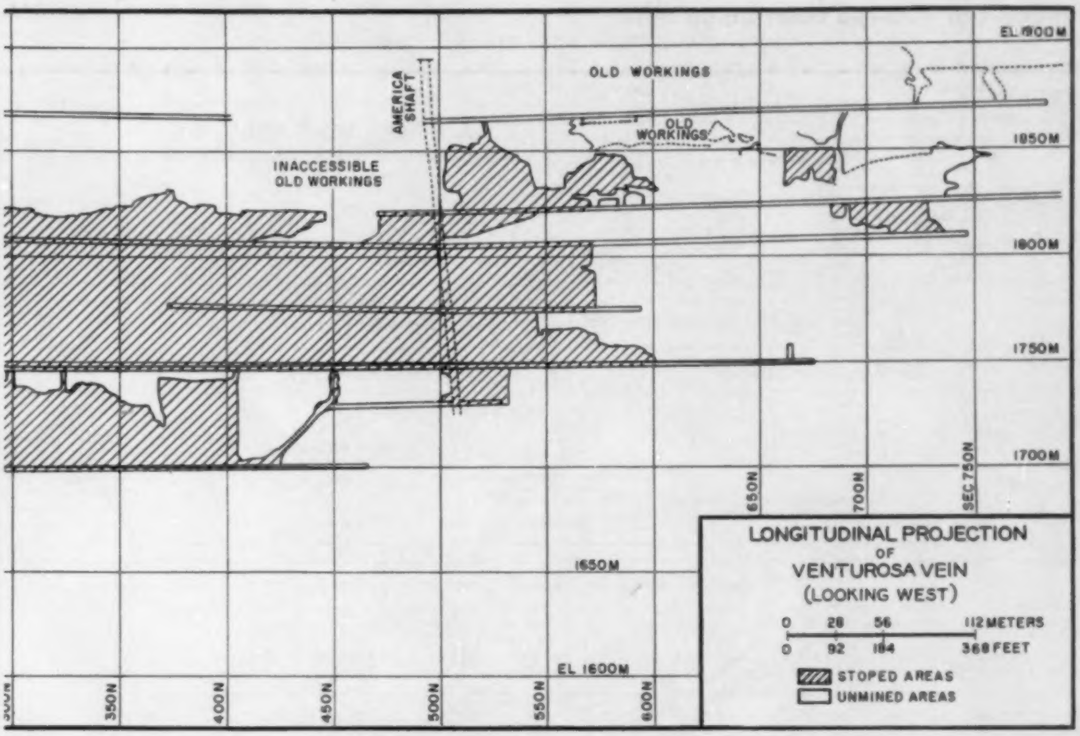
Shaft Sinking and Development: A four-compartment vertical shaft was started in August 1947, so that the newly found deep orebodies could be rapidly and economically mined. A temporary wooden head frame was erected and 60 ft of shaft was sunk through the surface wash. The shaft was cemented to a depth of 48 ft. Sinking continued rapidly to a depth of about 90 meters or 295 ft, where ground water in quantity forced many delays. A Riddell shaft mucking machine proved to be of great value in the extremely wet shaft.

Large stations were cut on the 100, 150, and 200-meter levels (approximately 325, 500, and 750-ft leads) and the shaft was sunk to a depth of 209 meters (686 ft) by the end of 1948. In 1951 the shaft was completed to a final depth of 265 meters (870 ft) and a station was cut on the 250-meter level. Large pump stations and sumps were cut on each level and crosscuts were driven to the veins. On July 17, 1948, the crosscut on the 100-meter level from the General shaft reached the Venturosa vein. Ore was first hoisted the following day and was stockpiled on surface until the mill began operating in September.

Orderly development in the mine was hindered throughout the life of the mine by the extremely large flows of water in the lower levels, because, to hold down the pumping cost, not too much ground could be opened ahead of actual mining. An extensive system of water doors was installed to control the water flow, but there were still delays.

Some six miles of underground workings were driven and an additional four miles of old workings reopened and retimbered.

Mining Methods and Development: The two principal veins at Namiquipa averaged more than 2 meters (6½ ft) in width, although in places greater widths were attained. The steep dips and the very solid hanging walls and footwalls of the veins made shrinkage stoping the ideal mining method. In the lower grade portions of the veins pillars of ore were left above the levels between stopes and chutes.



Where the veins were high grade, stoping operations were carried from the drift level. Drift sets of native round timber were used as support under the stopes. Chutes were placed at 4½-meter (15-ft) intervals.

Ore mining was hampered by large lenses of unaltered andesite found in the vein. These were probably rock-horses broken from the walls but not completely altered by ore-bearing solutions. As no ore sorting could be done, these rock-horses were either mined or left as pillars, depending on conditions. Ore in stopes was broken with pneumatic stopers. Production averaged 20 tons per machine shift during the life of the mine. In the widest part of the main vein experimental blastholes were diamond-drilled and blasted as the ore was needed. More than 10,000 tons of ore were broken in the experiment.

Since 1948 a total of 736,756 dry metric tons (820,000 short tons) of ore was produced, containing some 330 metric tons (367 short tons) of silver, 25,000 metric tons (28,000 short tons) of lead and 37,000 metric tons (41,000 short tons) of zinc, as summarized in accompanying tables. Gold production was less than 2200 lb. No accurate production figures prior to 1948 are available, but stoped areas and tailings piles on surface indicate that earlier production had probably been about 150,000 tons, making the total production from the property some 900,000 metric tons (1 million short tons).

The total of 736,756 metric tons (820,000 short tons) of ore milled since 1948 compares with the estimate of 800,000 tons indicated by diamond drilling only, an estimate made prior to the start of underground development. High taxes and rapidly increasing pumping costs prevented any consideration of sinking one or two more levels, from which it was believed additional ore might have been mined, low in silver and lead, but higher in zinc.

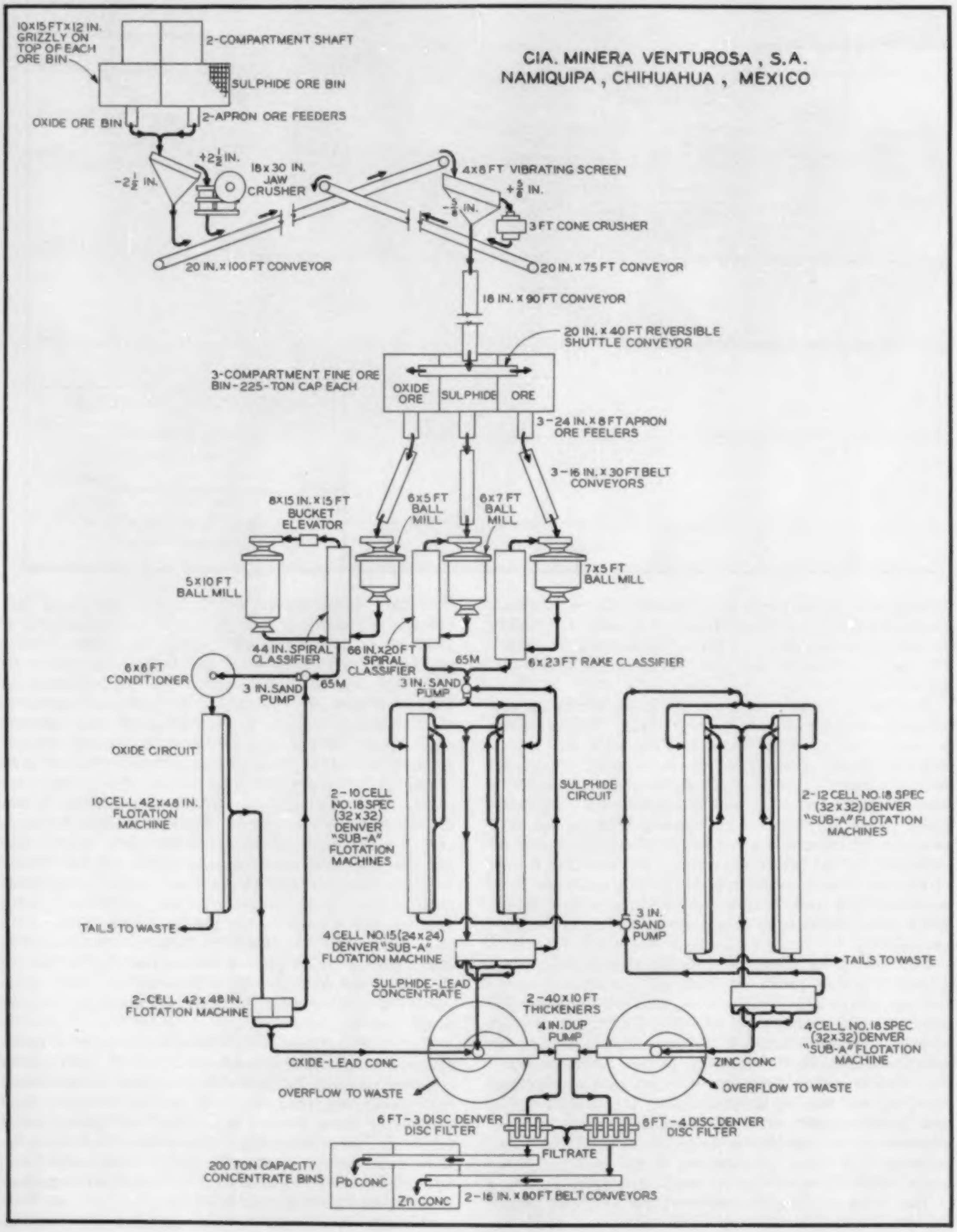
Water: Some water in the mine had been expected, but nothing like the flow that increased to 4500 gpm, averaging 4200 gpm when the bottom levels were abandoned in 1955. Direct pumping cost since the start of operations was $1.2 million in U. S. currency, an average of $14,390 per month, or $1.62 per ton milled. It is estimated that indirect costs incurred because of excessive water underground added at least an equal amount for such items as lowered efficiency of the crews working under adverse conditions; delays in exploration and development work; greatly increased cost of repairs and maintenance of the motors, cars, drills, and other equipment used underground; rubber clothing and boots for the miners; and probably of most importance, large investment in additional diesel engines and pumps to handle the water. More than two thirds of the diesel units were installed because of the great flow of water. During the life of the operation probably 13 billion gal of water were pumped, an average of 88 tons of water for each ton of ore hoisted.

The most serious stoppage of underground operations because of water occured July 9, 1954, when a lightning bolt burned out the cable in the shaft and paralyzed pumping for several hours. The 250-meter level flooded and remained under water for more than a month. More than two months were required to restore normal mining conditions, and production was severely restricted during July, August, and part of September.

Milling: On March 1, 1948, ground was broken for a modern flotation mill, and on September 18th the crushing plant began to operate on oxide ore. Operation of the oxide section of the mill started September 28th and operation of the sulfide section November 24th.

A complete metallurgical and assay laboratory was installed in the mill office building. Metallur-

Flowsheet of 500-tpd Namiquipa mill



Built in 1948, the modern 500-tpd flotation plant incorporated both sulfide and oxide sections. Separation of the two ore types continued from the divided bin throughout the plant to final concentrate thickening section. Plant production was about 10 million oz Ag, 55 million lb Pb, and 81 million lb Zn. Primary metallurgical problems in operation involved maintaining adequately high zinc concentrate grade, as well as preventing zinc losses into the lead concentrate.

Metallurgical Results of the Venturosa Operation

Oxide Section	Dry Metric Tons	Wt, Pct	Assays Ag, Grams	Assays Pb, Pct	Assays Zn, Pct	Distribution Ag	Distribution Pb	Distribution Zn
Heads	214,127.00	100.00	567	2.59	—	100.00	100.00	—
Lead concentrate	14,837.97	6.92	6898	24.10	—	84.28	64.32	—
Tailing	199,289.03	93.08	96	0.99	—	15.72	35.68	—
Sulfide Section								
Heads	522,629.00	100.00	401	3.76	7.07	100.00	100.00	100.00
Lead concentrate	31,863.37	6.10	4752	49.45	14.17	72.20	80.11	12.23
Zinc concentrate	46,117.91	8.82	447	1.87	52.25	9.83	4.39	65.23
Tailing	444,647.72	85.08	85	0.68	1.87	17.97	15.50	22.54

Principal Development Headings and Advances in Meters by Years

	1947	1948	1949	1950	1951	1952	1953	1954	1955
Main shaft	65.5	142.5			56.7				
Drifting		344.0	1164.4	1227.8	436.5	541.7	517.7	311.7	247.2
Crosscutting		265.0	333.2	179.0	544.0	236.0	48.4	40.9	
Raises and winzes			357.6		63.5	113.2	219.4	95.8	120.0
Pump stations				41.5					4.0
Ventillation shaft		50.0							
North shaft sinking				11.8	39.8				
American shaft sinking		41.0			23.0			18.0	
Total meters	65.5	842.5	1855.2	1460.1	1163.5	890.9	785.5	466.4	371.2

Distribution of Power Plant Load, 1948 Through 1955

Year	Total Kilowatt Hours Produced	Mine Pumps Kilowatt Hours	Mine Pumps Percent of Total	Compressed Air Kilowatt Hours	Compressed Air Percent of Total	Mine General Kilowatt Hours	Mine General Percent of Total	Mill General Kilowatt Hours	Mill General Percent of Total	Service and Light Kilowatt Hours	Service and Light Percent of Total
1955	8,988,380	5,573,180	62.00	712,353	7.93	188,080	2.09	2,168,559	24.13	346,190	3.85
1954	17,037,220	12,061,213	70.79	769,919	4.52	323,440	1.90	3,391,440	19.91	491,206	2.88
1953	17,958,870	11,984,567	66.73	1,204,230	6.71	323,293	1.80	3,959,272	22.05	487,508	2.71
1952	17,438,770	11,083,460	63.56	1,540,524	8.83	380,296	2.18	3,965,308	22.74	469,182	2.69
1951	15,692,410	9,448,633	60.21	1,721,545	10.97	296,056	1.89	3,669,914	23.39	556,262	3.54
1950	11,715,350	6,011,757	51.32	1,789,860	15.28	321,795	2.75	3,067,598	26.18	524,340	4.47
1949	7,963,797	3,071,057	38.57	1,669,415	20.96	344,328	4.32	2,576,909	32.36	302,088	3.79
1948	646,637	116,306	18.00	270,341	41.81	91,695	14.18	145,351	22.48	22,864	3.53

gical testing units of 2000 and 500-g capacity were used. Laboratory pressure filters, a test ball mill capable of grinding 2000-g samples, and testing sieves completed the equipment. Samples were prepared first in a small crusher and then in a Braun pulverizer.

Crushing Section: Ore from the mine was hoisted in cars and dumped in a bin divided into two parts, one for the sulfide and the other for oxide ores. From the bin the ore was fed by apron feeders to the primary crusher, where it was crushed to about 2½ in. It then fell to an inclined conveyor that discharged it to a vibrating screen. The mesh of the wire screen governed the size of ball mill feed, the undersize falling to a conveyor belt going to the fine ore bins and the oversize into a chute to the secondary crusher. The product of this crusher was returned to the conveyor handling the product of the primary crusher so that it, again, passed over the same vibrating screen. A metric weightometer on the conveyor belt weighed the crushed ore on its way from the crushing plant to the fine ore bins.

Grinding Section: Three ball mills were used regularly—a No. 75 Marcy mill and two Allis-Chalmers mills, one 6-in. diam and 5 in. long and the other 6-in. diam and 7 in. long. Each mill operated in a closed circuit with its classifier—a Western Machinery Co. spiral classifier for the No. 1 and No. 2 mills; a Dorr model D for the No. 3 mill; and a Dorr bowl classifier, type DSFB, for the small spare No. 4 mill. At times more than 500 tpd were milled.

Flotation Section: Flotation equipment in the oxide section consisted of a 6x6-in. conditioner, a 10-cell flotation machine for roughing, and two cells of the same type for cleaning. Flotation equipment for sulfide ore consisted of two 10-cell machines for lead and two 12-cell machines of the same size and type for zinc. There was also a 2-cell machine for zinc recleaning and six concrete tanks, equipped with agitating mechanisms, for dissolving flotation reagents.

Thickening and Filtering: There were two thickeners, 40 ft diam and 10 ft high, one for lead concentrate from both oxide and sulfide sections and the other for zinc concentrate. Pulp was sent from thickeners to filters by a diaphragm pump. Filtered concentrates fell to conveyor belts, which transported them to their respective bins.

Metallurgy: The Venturosa tonnage of ore yielded in recoverable contents 10,609,700 oz of silver, 55,611,500 lb of lead, and 81,428,600 lb of zinc.

The lead collector that gave consistently good results was Aero-float reagent No. 242, which had to be added in extremely small amounts to avoid floating the sphalerite. It was found that adding more than 0.25 lb of cyanide per ton depressed silver, the most valuable constituent of the ore.

The two chief metallurgical problems were to obtain a zinc concentrate of salable grade and prevent loss of zinc values in the lead concentrate. Apart from the loss of zinc into the lead concentrate there was the further problem of the zinc feed being so impoverished in zinc and the zinc being so strongly depressed that it became difficult to achieve good results in the zinc section. To a considerable degree these difficulties were overcome by crowding the froth in the cleaners.

Power: Earlier operators at Namiquipa used wood-burning steam turbine plants to generate electric power. The Venturosa Co. installed two

Sulfide Production, 1948 Through 1955

Year	Tons	Grade: Ag, Grams	Grade: Pb, Pct	Grade: Zn, Pct	Contents in Metric Tons: Ag	Contents in Metric Tons: Pb	Contents in Metric Tons: Zn
1948	5,346	1,079	9.66	12.76	5.768	516.6	682.5
1949	62,151	541	5.23	8.21	33.624	3,252.1	5,101.7
1950	82,209	562	4.43	8.22	46.201	3,643.7	6,754.3
1951	89,105	394	2.92	6.73	35.107	2,601.5	5,997.8
1952	91,021	317	3.87	7.22	28.854	3,518.1	6,571.3
1953	89,245	359	3.46	6.63	32.039	3,086.9	5,913.5
1954	76,960	309	3.05	5.82	23.781	2,347.5	4,480.7
1955	26,592	163	2.64	5.41	4.335	702.6	1,437.7
Total	522,629	401	3.76	7.07	209.709	19,669.0	36,939.5

Oxide Production, 1948 Through 1955

Year	Tons	Grade: Ag, Grams	Grade: Lead, Pct	Contents in Metric Tons: Ag	Contents in Metric Tons: Pb
1948	6,468	624	3.74	4.036	241.9
1949	21,119	791	3.50	16.705	739.0
1950	29,682	947	3.23	28.109	957.5
1951	38,306	661	2.57	25.320	986.1
1952	36,963	581	2.50	21.476	923.4
1953	27,909	376	3.09	10.494	861.1
1954	13,133	301	2.75	3.953	361.6
1955	40,547	280	1.20	11.353	485.8
Total	214,127	567	2.59	121.446	5,556.4

Combined Oxide and Sulfide Production, 1948 Through Aug. 31, 1955

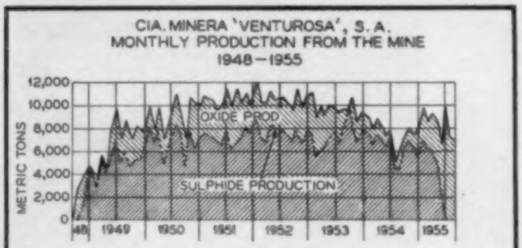
Year	Total Metric Tons Milled	Average Grades: Ag, Grams	Average Grades: Pb, Pct	Average Grades: Zn, Pct	Contents in Metric Tons: Ag	Contents in Metric Tons: Pb	Contents in Metric Tons: Zn
1948	11,814	830	6.42	5.78	9.8	758.5	682.5
1949	83,270	604	4.79	6.13	50.3	3,990.9	5,101.7
1950	111,891	664	4.11	6.04	74.3	4,601.3	6,754.3
1951	127,411	474	2.82	4.71	60.4	3,587.6	5,997.8
1952	127,984	394	3.47	5.13	50.3	4,441.5	6,571.3
1953	117,154	363	3.37	5.05	42.6	3,948.0	5,913.5
1954	90,093	308	3.01	4.97	27.8	2,709.0	4,480.7
Through Aug. 31, 1955	67,139	214	1.77	2.14	15.7	1,188.6	1,437.7
Total	736,756	448	3.42	5.01	331.2	25,225.4	36,939.5

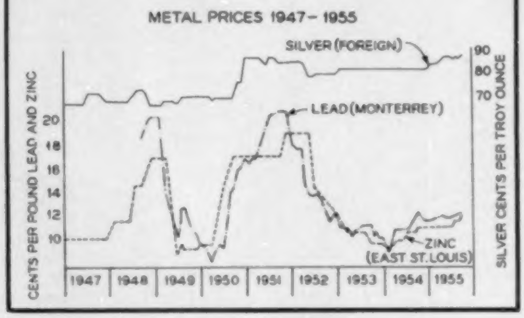
Reagents Used in Oxide Section

Reagent	Average Pounds Per Ton	Fed At:
Sodium sulfide	0.808	Ball mill
Sodium sulfide	0.100	Rougher cells
Z-6 xanthate	0.245	Flotation feed
Aerofloat No. 404	0.285	Flotation feed
Aerofloat No. 242	0.007	Flotation feed
Aerofloat No. 208	0.116	Flotation feed
Pine oil	0.005	Flotation feed
Frother	0.024	Flotation feed

Reagents Used in Sulfide Section

Reagent	Average Pounds Per Ton	Fed At:
	Lead	
Zinc sulfate	1.042	Ball mill
Sodium cyanide	0.257	Ball mill
Aerofloat 242	0.076	Flotation feed
Frother B-23	0.010	Flotation feed
	Zinc	
Copper sulfate	1.488	Flotation feed
Sodium Aerofloat 'B'	0.111	Flotation feed
Hydrated lime	0.742	Flotation feed
Z-3 xanthate	0.079	Flotation feed
Frother	0.034	Flotation feed
Sodium cyanide	0.111	Zinc recleaner





small diesel engines in 1947. By 1955 there were nine diesel engines of various sizes with a total rated 8266 brake-horsepower. Power generated at 2300 v, 60 cycles, was distributed by overhead lines to the mine, mill, shops, and dwellings, where the voltage was reduced. An underground three-conductor electric cable laid between the power plant and the mine transformer station could be used independently or in conjunction with overhead lines.

A tank farm with a capacity of more than a million liters of fuel oil was erected at Namiquipa. Fuel oil was transported in a fleet of tank trucks running between the railroad siding at Tejolocachic and the mine. Some 15,000 liters (about 4000 gal) of oil were burned in an average day's operation.

Costs and Taxes: From the time the Venturosa Co. started development the conditions under which the Mexican mining industry operated grew progressively worse, owing to successive devaluations in the peso and a greatly increased burden of taxation of all kinds. When the operation started the exchange rate was 4.85 pesos to one U. S. dollar. At the close of the operation it was 12.50 pesos to one U. S. dollar. Any expected advantage in a cheaper peso was more than offset by increases in taxes, wage scales, and prices of materials bought in Mexico.

During that same period an *ad valorem* export tax was imposed that applied to all metals produced by the company. The initial rate, about 15.3 pct, was later raised to 25.5 pct on silver, 28.05 pct on lead, and 28.305 pct on zinc. Production tax rates were also automatically increased as a result of each devaluation of the peso, as such tax rates are based on the Mexican currency equivalent of the U. S. price. These production and export taxes, the heaviest of the tax burdens imposed on the mining industry, were based on gross income and therefore payable whether or not the producer realized a profit. At the time the Venturosa Co. ceased operating in 1955, these taxes alone took 35 pct of the gross value of the silver produced, 30 pct of the lead, and 13.5 pct of the zinc value. Any

The two principal veins at Namiquipa proved ideal for shrinkage stoping, averaging more than 6 ft wide. In this stope view ore is being drilled with stopers working off the muck pile.

Battery locomotives were used for haulage on all levels, pulling side-dump cars of about 2-ton capacity. Mechanical loading was used wherever possible.

profit remaining after the payment of these taxes on gross income and operating expenses was then subject to the regular income, dividend, and excess profit taxes, which also increased substantially during this period. The regular income tax was increased to a top bracket of 39 pct on all profits higher than the equivalent of $160,000 in U. S. currency. All profits exceeding 15 pct of invested capital were made subject to an excess profits tax, at rates up to 25 pct on profits higher than 50 pct of invested capital, and were also subject to certain limitations recently enacted. The so-called dividend tax, or the tax on distributable profits, was increased from 10 to 15 pct.

The effect of such combined taxes, particularly those on gross income, may be seen from the following figures. From the start of operations in 1948 to the end of 1955 the company's profit before all taxes amounted to $4,652,853.63 in U. S. currency, including the very generous subsidies of $893,446.57. After the total net taxes of $4,514,556.98, the company's net income for the period was $138,296.65.

The steadily improving domestic economy of Mexico appears to be of such importance to the rising total tax yield that the Government probably will be able to lighten substantially the burden of taxation on the mining industry. Some steps in that direction have already been taken.

Because of the difficult physical operating conditions and the tax situation, continuous and concerted efforts were made to control all operating costs.

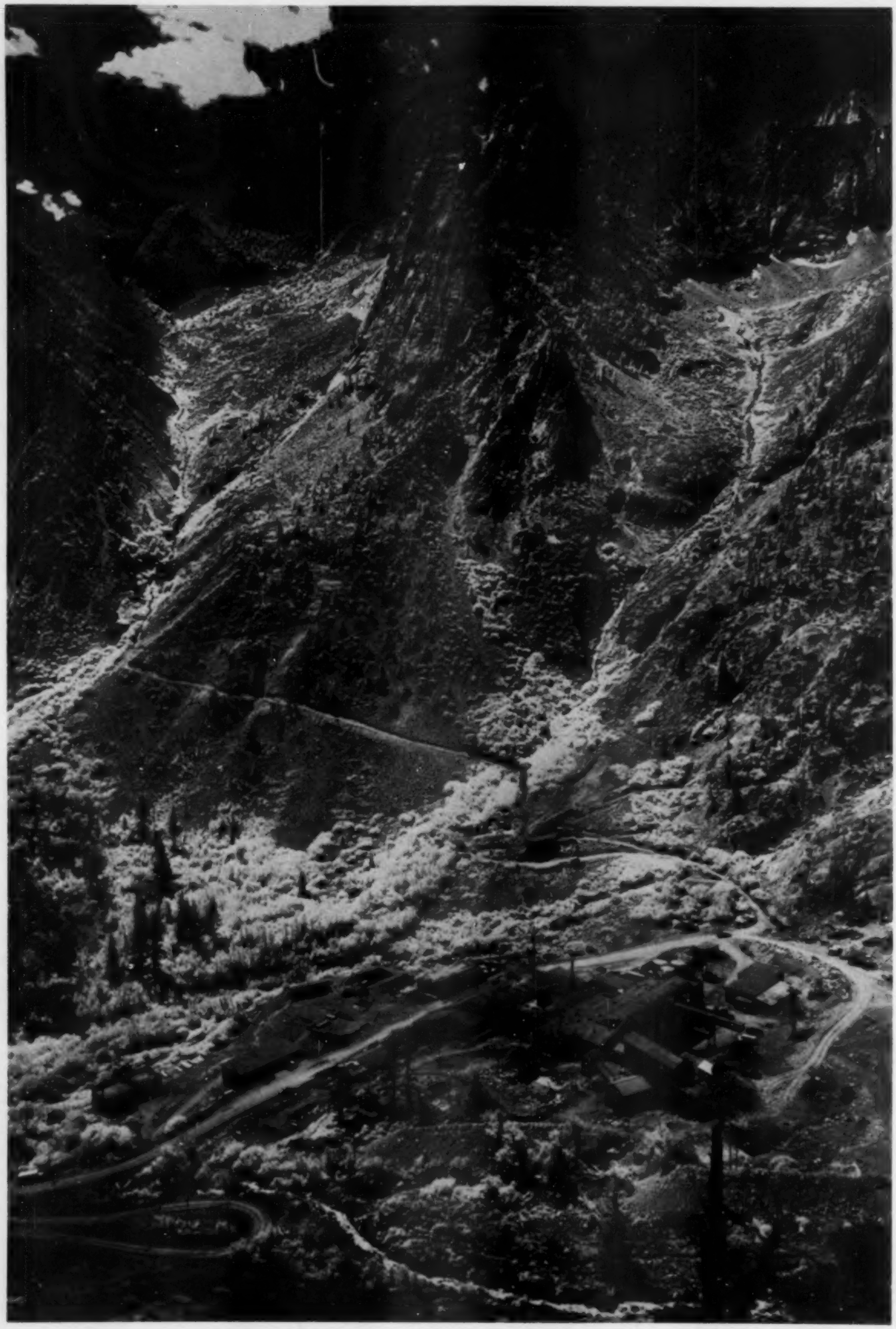
Costs in U. S. Currency Per Metric Ton Milled, 1948 Through 1954

Year	Costs Other Than Pumping, $	Direct Pumping Costs, $	Total Costs Before Depreciation, Exchange Adjustments, and Income Taxes, $
1948	11.53	0.68	12.21
1949	10.52	1.31	11.83
1950	9.72	1.11	10.83
1951	9.09	1.64	10.73
1952	8.88	2.00	10.88
1953	7.95	1.66	9.61
1954	7.70	2.22	9.92

Tons Per Man Shift: The overall effectiveness of the operation can be appreciated from the fact that an average of 0.95 tons was milled per manshift worked both on surface and underground during the period from 1951 to 1955, outstanding for a Mexican operation of this type of mine.

Despite mining difficulties and high taxation, the cost of opening and equipping the property—some $1.85 million—was repaid with interest of approximately $300,000. Upon final liquidation further distribution is expected.

About 400 men were employed by Cia. Minera Venturosa, S. A., at Namiquipa when operations were in full swing. Large land areas were irrigated by water from the mine. Good miners were developed from among the people of the countryside. Miners, mechanics, accountants, mine surveyors, and engineers were trained, and this in itself was an important contribution to the economy of the area.



Mining Methods at Pine Creek Mine

by H. L. McKinley and L. A. Wright

PINE Creek mine of Union Carbide Nuclear Co. is some 23 miles northwest of Bishop, Calif., in the Sierra Nevada Mts. Office and mill are 7800 ft above sea level, the 1500 level portal is at 9300 ft, the A level portal at 10,800 ft, and the 3050 portal at 11,500 ft.

Original claims were located in 1916, and the mine worked intermittently until 1936, when it was purchased by Union Carbide Corp. Since then the property has been operated continuously, except for two brief periods when operations were suspended because of manpower shortages. Mine production through 1956 amounted to 2,792,425 dry tons of ore.

The orebody of Pine Creek mine is a contact-metamorphic zone between quartz-monzonite and limestone. The limestone has been altered to a tactite, in which the ore minerals occur. The main ore mineral is scheelite, which is finely disseminated throughout the tactite. Molybdenite, chalcopyrite, and bornite are present in the tactite as small stringers and are occasionally disseminated.

The contact zone dips vertically, and rock alteration has produced a hard, dense area suited to the open stope method, so there has been no caving of stope walls.

Since the ore shoots are dispersed along the contact zone and vary in horizontal and vertical dimensions, developing and prospecting the ore shoots entails considerable waste work and duplication of development excavations.

The system of mining in use can be described as *sublevel blasthole stoping*. This method, which is adapted to the ore shoots that are 20 ft wide or more, produces most of the tonnage. For ore shoots less than 20 ft wide conventional shrink stoping is employed. In general, development and preparation excavations produce 32 pct of the tonnage broken, blasthole mining produces 62 pct, and shrink stoping 6 pct.

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Mine Development: A 7000-ft haulage tunnel 9430 ft above sea level is the main entry to Pine Creek mine. Most of its length is in granite. All mining activities are conducted above this elevation and the broken ore is transferred to this level by gravity. The tunnel is 9x12 ft in cross section, the extra width allowing for a water ditch that drains the mine areas at a rate of 4000 gpm. Two 13-ton trolley-electric locomotives on 250-v dc power haul all materials, ore, and men through this tunnel, traveling on track of 36-in. gage over an average grade of 0.6 pct. Power is supplied by two 75-kw motor-generator sets, one at each end of the line.

The rotary dump ore cars, of 185-ft capacity, average 11 tons per car and there are eight cars in a train. Men are transported in locally constructed man cars, and materials are transported on conventional flat cars. Production figures for ore haulage are as follows: tons per direct labor shift, 140; percent of direct labor hours to total mine labor hours, 8.6 pct; material cost per ton, 12 pct; labor 50 pct; and expense items 38 pct.

A 1585-ft vertical raise with two 6x6-ft compartments affords access to the working areas above.[1] The main hoist at the bottom of the raise is a single-drum hoist powered by a 125-ac motor. Installed in one of the raise compartments is a two-deck cage, built partly of aluminum, which carries 18 men. One-inch 6x19 hoisting cable is used.

Extending 800 to 2200 ft along the contact zone from the main service raise, there are five working levels spaced on 200-ft intervals and two working levels spaced on 250-ft intervals. Working levels, locally called main levels, consist of track drift 8x8 ft or 9x9 ft in cross section, with 18-in. track gage. The Granby-type ore or waste cars, of 85-cu ft capacity, are driven by 3-ton battery locomotives.

Main level drifts are driven at a rate of 3800 ft per year, with two or more levels advanced concurrently on a single or double shift basis. At each drift face a two-man crew mucks out, drills, and blasts a 5 to 7-ft round each shift. Drilling is carried

Two techniques have been used for blasthole stoping—sublevel drifts and undercuts with vertical drilling. Although not fully evaluated, it appears that cost per ton will be about the same for the two methods, but that undercutting, shown here, will offer greater flexibility and better results. Disadvantages of undercutting include added development time and added time to clean up the sill. Advantages include good fragmentation, reduction in core drilling for exploration ahead of mining, and some selectivity in mining.

out with two 3½-in. Leyners, using 1¼-in. round steel with 1⅝-in. detachable tungsten carbide bits. The Leyners are mounted on a two-boom drill jumbo, the booms actuated by compressed air. Eimco 21 mucking machines muck into the Granby cars. Car-pass stations, located at 200-ft intervals, are later used as diamond drill stations.

About 44 holes are needed to break a 9x9 cross section using 22 lb of 45 pct powder per ft. Advance per direct labor shift averages 1.45 ft, direct labor hours amounting to 16.4 pct of total mine labor hours. According to a cost breakdown per foot, material is 29 pct, labor 40 pct, and expenses 31 pct.

Main level drifts are driven for two primary purposes, for prospecting and for access to orebodies. These drifts are driven parallel to the contact zone and 40 to 60 ft away from it, usually in granite. This location of the drift permits periodic drilling during drift progress to prospect the contact zone and will eventually permit extraction of ore left in the main level pillars without destroying the main level drift.

All diamond drillholes are AX size, and core recoveries exceed 95 pct. The geology and engineering departments control the location, direction, and spacing of the coreholes and are responsible for logging and sampling the cores. The first program of drilling uses a wide spacing of holes, generally 100 to 150 ft apart, to locate the contact zone and determine the presence or absence of ore. If ore is encountered, the second drilling program spaces holes 50 to 75 ft apart to delineate the ore in horizontial and vertical dimensions and to determine ore grade. Closer spacing of drillholes may or may not be required to delineate critical areas or to sample the ore for grade determinations in borderline ore shoots. There is no additional sampling of the orebodies.

The orebodies thus defined by diamond drilling from the main level drifts are assigned a block number and the entire development of the block is planned before work begins. Block development then proceeds from the main level drifts by crosscutting to the ore zone.

Block Development: Two units of the block development are usually started and advanced concurrently—the service raise and the scraper drift or grizzly.

Service raises are usually driven vertically and are located on the edge of the ore and in the center longitudinally of the block. The raises are timbered with four post sets that are one compartment of 5x5-ft size. Rock excavation is 7x11 ft, so that the muck compartment of the raise is composed of lagging applied to the timber sets on one side and the bare rock on three sides. A two-man crew works at the face of the raise, one man operating the tugger hoist from the bottom. A timber boat is installed (and operated by the tugger hoist) which operates through the raise on lagging acting as a slide. Drilling equipment consists of two 3½-in. stopers, using 1-in. Hex steel with 1 ⅝-in. detachable tungsten carbide bits.

The two-man crew timbers and sets up the stopers on one shift and drills, tears down, and blasts a 5½-ft round the next shift. About 55 holes are needed to break the 7x11-ft cross section, requiring 24 lb of 45 pct powder per ft. Advance per direct labor shift averages 0.65 ft, direct labor hours averaging 7.1 pct of total mine labor hours. The four post timber sets and related timber cost $13 per ft —25 pct for materials, 44 pct for labor, and 31 pct for expense items.

Scraper drifts are generally used for ore disposal; a system of grizzlies is employed only in isolated locations or in irregularly shaped orebodies where a straight scraper drift cannot be adapted. Both the scraper drift and the grizzly system discharge the ore into open stopes or into chutes for car haulage.

The scraper drifts are 9x9 ft in cross section, with maximum scraping distance of 100 ft. At 25-ft center-to-center spacing, a 7x7-ft side set is driven at right angles to each scraper drift. From these side sets 7x7-ft untimbered raises are driven on 50° slopes to the desired limits and are connected at their maximum elevation, which is usually 25 ft above the back of the scraper drift. Rock between the connected raises in mined out (belled), producing an inverted cone excavation above each side set, the base 20 to 25 ft in diam and the height 20 ft. This same type of excavation is done from the side sets leading to grizzly chambers. When the belling is completed the grizzly or scraper unit is ready for stoping. A 50-hp two-drum slusher is installed in the scraper drift, together with a 48-in. quarter box scraper.

The scraper or grizzly drift, side sets, and chambers are driven by jackleg machines using Copco integral alloy steel. The 3½ in. stopers are used for untimbered raises and belling. Combined advance per direct labor shift is 1.37 ft, direct labor hours averaging 5.1 pct of total mine labor hours. Material accounts for 27 pct of the cost per foot, labor 45 pct, and expense items 30 pct.

To aid ventilation and to ease the supply problem, sublevel drifts are not driven from the service raise

Main entry to the property is provided by a 7000-ft tunnel that is largely driven through granite. The tunnel portal is more than 9000 ft above sea level. The 9x12-ft cross section provides drainage for 4000 gpm, as well as accommodating the 13-ton trolley locomotives on 36-in. gage track. Two 74-kw motor-generator sets supply 250-v dc power for haulage. Average grade is 0.6 pct. Men are transported in locally built cars and ore is moved in 185-cu ft rotary dump cars carrying an average of 11 tpd per car.

until the raise is completed by holing into the next main level drift above. The sublevel drifts are driven from the service raise at vertical intervals of 35 to 45 ft. The broken ore produced is discharged to the scraper drift belling by an ore pass raise. Sublevel drifts, 9x9 ft in cross section, are usually driven both ways from the centrally located service raise to the extremities of the orebody—a maximum of 130 ft from the raise.

The drifts are driven by jackleg machines and 20-hp two-drum slushers with a 42-in. quarter box scraper. The two-man crew advances the face 5 to 7 ft each shift. Average advance per direct labor shift is 0.95 ft, direct labor hours averaging 7.9 pct of total mine labor hours. According to cost breakdown per foot, material is 19 pct, labor 47 pct, and expense items 34 pct.

Another method of blasthole development used instead of the sublevel drift is called an *undercut*, a local name applied to an excavation that is 9 ft high and excavated to the total width and length of the orebody. Spaced at the same vertical intervals as the sublevel drifts, undercuts are used for blasthole stoping. Parallel vertical blastholes are drilled.

Jackleg machines and 20-hp two-drum slushers are used in excavating undercuts. Average advance per direct labor shift is 151 cu ft, direct labor hours averaging 12.8 pct of total mine labor hours. Material is 20 pct of the cost per cubic foot, labor 47 pct, and expense items 33 pct.

At present Union Carbide Nuclear Co. is experimenting with an Air Trac type of machine in excavating undercuts. The Air Trac is being used as a two-machine jumbo, with two 3½-in. Leyners mounted on a crossbar. A 20 pct reduction in undercutting costs should be possible.

To speed block development work ore pass raises are located in the most advantageous positions within an ore block and are also used as slot raises in blasthole stoping. These raises are usually vertical, maximum length 40 ft, and are driven untimbered. Drill staging is either removed or blasted out with each round, and a system of steel pins is driven into the ribs of the raise to aid access after each round.

Ore pass raises are 7x7 ft in cross section and driven by 3½-in. stopers using 1-in. Hex steel with 1⅝-in. detachable tungsten carbide bits. Average advance per direct labor shift is 1.70 ft, direct labor hours averaging 4.2 pct of total mine labor hours. Material is 23 pct of cost per foot, labor 45 pct, and expense items 32 pct.

Three ore pass raises have been excavated recently by drilling blastholes and blasting 5-ft sections of the hole pattern. One of these raises was drilled in soft ground and had to be abandoned, but the other two were very successful. This method of driving untimbered raises will be expanded in the future.

Intermediate haulage on main levels is required in a number of locations because the orebodies are dispersed. The 85-cu ft Granby cars are loaded by direct discharge from scraper drifts or by chutes from scraper drifts or grizzlies. Average tramming distance for this work has been 850 ft one way. Average production has been 47 tons per direct labor shift, direct labor hours averaging 21.5 pct of total mine labor hours. Material cost is 16 pct of the cost per ton, labor 49 pct, and expense items 35 pct.

Shrink Stope Mining: Shrink stoping in the Pine Creek mine is usually applied to the orebodies less than 20 ft wide and to the orebodies that are irregular in dip. Experience has shown that drawing ore from the stope at vertical distances of 90 ft or more above the drawpoints creates pockets in the broken ore column, presenting a safety problem and making it very difficult to control the level of the broken ore at the mining face.

Block development for shrink stopes consists of a scraper drift or a grizzly system and service raise. Service raise stations are at 25-ft intervals and are connected by untimbered raises, which are driven on the raise pillar line or on the limit of the ore shoot. Service to the shrink stope from the raise stations is through the untimbered raise, which is mined out as the stope back is mined.

Actual mining of the stope back is done by horizontal cuts taken successively from one end of the stope toward the other end, with a vertical face on each cut of about 12 ft. Jackleg machines are used with integral steel. Average performance, including drawing the broken ore, is 30 tons per direct labor shift, direct labor hours averaging 2.2 pct of total mine labor hours. Material is 21 pct of cost per ton, labor 46 pct, and expense items 33 pct.

Blasthole Mining: Blasthole mining is safe, flexible, and inexpensive and is particularly suited to the physical character of the Pine Creek mine orebodies. Two types of blasthole mining are used—ring drilling and parallel vertical holes.

Rings are drilled at right angles from sublevel drifts, which are driven parallel to the strike of the ore shoot. All rings have been either a 90° or a 180° pattern. The 90° pattern starts with a horizontal hole and the remaining holes are spaced downward

Orebodies are defined by diamond drilling from the main level drifts and are assigned a block number. Initial drilling with wide-spaced holes locates the contact zone, followed by closer drilling to define horizontal and vertical dimensions of any orebodies found.

Direct Labor Hour and Cost Distribution

Operation	Direct Labor Hours to Total Mine Labor Hours, Pct	Cost in Percent of Total: Material	Labor	Expense	Total All
Main level drifts	16.4	22.6	18.0	19.3	19.6
Diamond drilling	4.0	15.9	3.7	3.9	6.8
Scraper and grizzly	5.1	5.9	5.4	5.3	5.5
Undercutting	12.8	8.8	11.9	11.8	11.1
Service raises	7.1	7.0	7.3	7.2	7.2
Sublevel drifts	7.9	5.2	7.3	7.2	6.7
Untimbered raises	4.2	3.6	4.1	4.2	4.0
Blasthole drilling	8.9	13.4	8.4	7.7	9.5
Shrink stope	2.2	1.6	2.1	2.1	1.9
Intermediate haulage	21.5	11.1	20.1	20.4	18.0
Main level haulage	8.6	4.6	10.6	10.9	9.2
Waste haulage	1.3	0.3	1.1	0.0	0.5
	100.0	100.0	100.0	100.0	100.0

Unit Break Down of Costs

Operation	Percent of Cost to Total Cost
Exploration and mine development	26.4
Block development	34.5
Blasthole and shrink stoping	11.4
Haulage (combined)	27.7
	100.0

to a vertical hole. The 180° pattern starts with a horizontal hole and ends with a horizontal hole in the opposite direction, all other holes being spaced at proper intervals under the horizontal holes. All holes in either ring are drilled in a vertical plane, with rings spaced 5 ft apart. The burden between ends of successive holes varies from 6 to 8 ft.

Ring drilling at Pine Creek mine has averaged 1.3 tons broken per foot of hole, with a powder factor of 0.51 lb per ton. Although the operation has been successful, fragmentation has always been poor, which increases secondary breakage and reduces production, with resulting higher costs. In spite of a ten-year program of experimentation with ring patterns and blasting procedures, fragmentation is still far from desirable. Moreover, the holes drilled in a ring pattern are fanned from the drill point in the sublevel drift so that the collars of the holes are only a few inches apart. This means that 35 pct or more of the total drill footage in each ring is not used.

Another big disadvantage of ring patterns in the Pine Creek mine is that the horizontal depth of any pattern cannot be expanded beyond the depth of the patterns drilled and blasted ahead to mine a *bulge* in the ore shoot walls. Detailed core drilling is also required on each sublevel to obtain the geological data necessary to design the ring pattern.

When parallel vertical holes are drilled, the ore at each sublevel elevation is mined to the full width of the ore shoot, vertical height of the excavation being 9 to 10 ft. Parallel vertical holes are then drilled from this undercut.

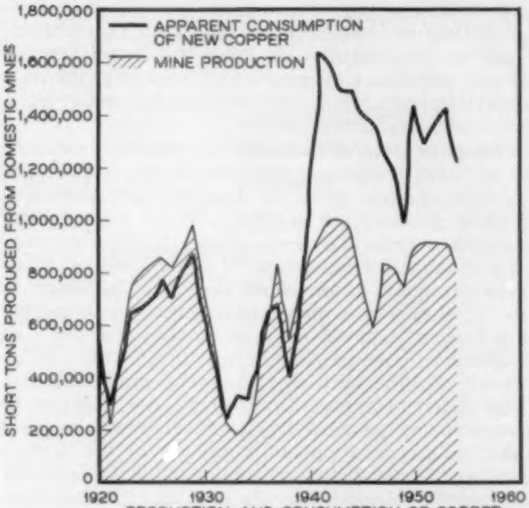
This method has averaged 2.2 tons broken per foot of hole, with a powder factor of 0.40 lb per ton. Holes are drilled in a staggered pattern similar to the patterns used in open pits and quarries. Fragmentation with this method has been very good, reducing secondary blasting and raising haulage production. Parallel vertical drilling offers increased flexibility in mining oddly shaped ore shoots and bulges, reduces waste dilution and eliminates wasted drill footage, reduces drilling time for the blasthole drilling operation, and reduces core drilling. There are some disadvantages—more development time is needed for the undercut than for the sublevel drift, and additional time and labor are required in cleaning up the sill of the undercut to check the area for missed lifters. There has not been enough operating experience for complete evaluation of the undercutting—parallel hole method as compared with the ring drilling system. At present it appears that cost per ton will be about the same, but results will be much better.

Undercutting also offers the possibility of major cost reduction in the immediate future and will permit the use of multiple machine jumbos in drilling blastholes.

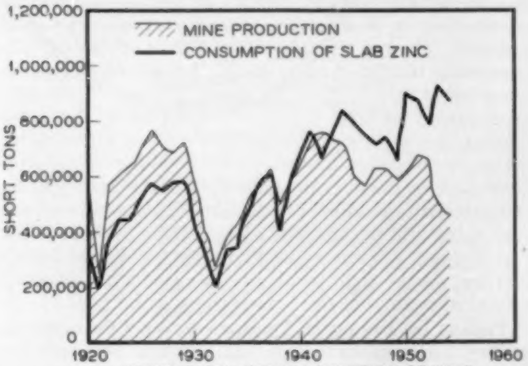
Blasthole drilling equipment consists of 4-in. percussion machines, using 4-ft sectional 1-in. Hex steel and 2¼-in. tungsten carbide bits. Drill footage averages 110 ft per machine per shift; 66 ft per direct labor shift, including moving and setting up; and 96 tons per direct labor shift. Direct labor hours average 8.9 pct of total labor hours. All blastholes are lined for their entire length with primer cord and the explosive is loaded to specified depths from a loading chart.

Reference

[1] J. F. Emerson and L. A. Wright: Method for Driving Long Service Raises, MINING ENGINEERING, April 1956, p. 399.



PRODUCTION AND CONSUMPTION OF COPPER IN THE UNITED STATES 1920–1954



PRODUCTION AND CONSUMPTION OF ZINC IN THE UNITED STATES 1920–1954

The Search for Mineral Raw Materials

by H. M. Bannerman

IN the past few years the mineral raw materials problem has risen from comparatively obscurity to great national significance. The transition has come so rapidly that the nature of the problem and what it portends have not yet been clearly defined. Consequently there are wide differences of opinion among the experts as to what can, or should be done to insure a continuing, adequate supply. On one hand, it is said that depletion of known domestic deposits and lagging discovery of new ones will bring about an imminent decline in mineral production and that in future U. S. mineral-using industries will have to depend on foreign sources for more and more of the major mineral raw materials. On the other hand, it is argued that domestic mineral resources are virtually inexhaustible if there were only legislation against competition from low-cost foreign producers. Still another school of thought contends that within the past decade the country has entered a period of unprecedented mineral discovery that is rapidly solving the problem of depletion.

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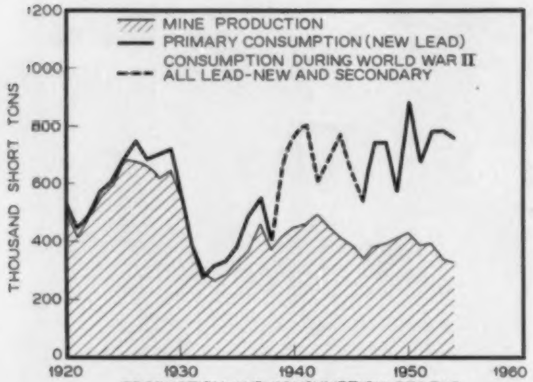
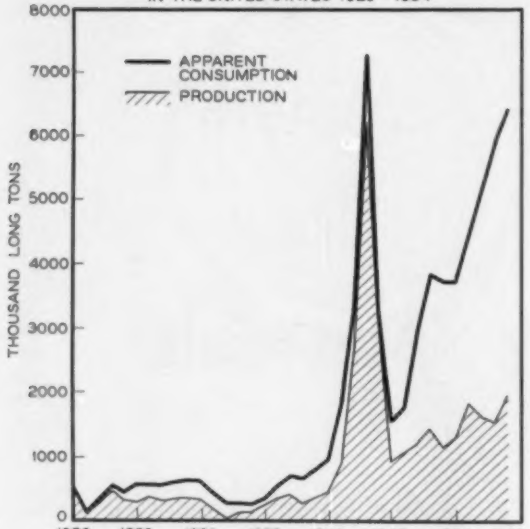
Only time will tell which of these widely divergent views is nearer the truth, for mineral resources are, after all, the result of research and adaptation, and resources of the future, as of the past, will depend largely on human response to circumstances. Of 18 substances now termed *reactive metals*, only 8 (boron, thorium, potassium, sodium, lithium, uranium, molybdenum, and tungsten) were considered of commercial significance at the turn of the century. Except for hafnium and rhenium, the other ten were well known to chemists and mineralogists before this, but none had found use commercially, whereas some, like titanium, were looked upon as nuisances in the ores of other metals. Of these ten elements, several were considered of minor commercial significance, and others were of little more than academic interest as recently as 15 years ago. Before World War II only a few of the better informed spoke of uranium or thorium as possible sources of power. Still fewer talked of converting boron into a superfuel, and the time had not yet arrived when hafnium, rhenium and even zirconium were thought of as critical to U. S. welfare and safety.

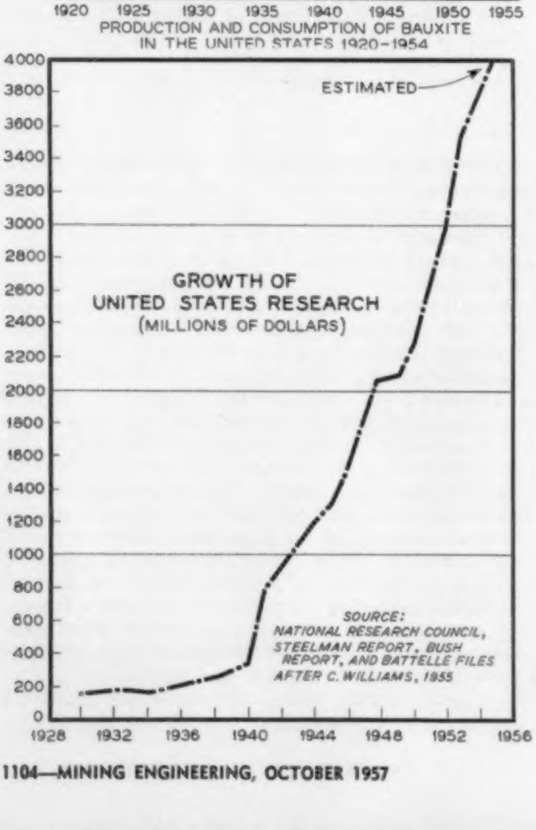
In other words, the importance of minerals changes with technology and with the times, and



PRODUCTION AND CONSUMPTION OF LEAD IN THE UNITED STATES 1920—1954



PRODUCTION AND CONSUMPTION OF BAUXITE IN THE UNITED STATES 1920-1954



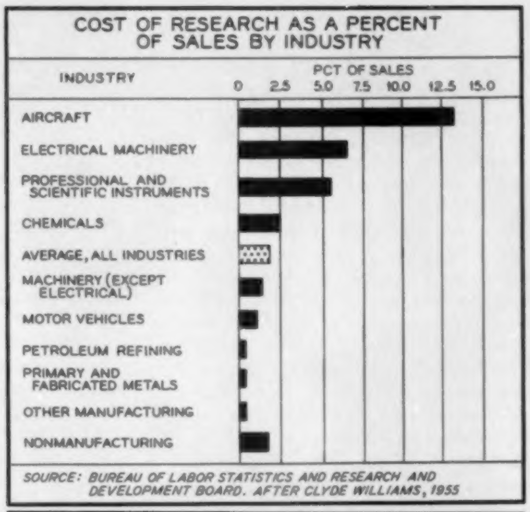
except as they are found useful to human enterprise, they are not mineral resources. This concept leads to the conclusion that the end is not yet, for by this premise, assuming a free economy, the only limitation that can be set on mineral resources is man's own resourcefulness.

Nature of Mineral Deposits: Mineral *resources* may be the product of man's resourcefulness, but mineral *deposits* are the result of long geologic processes. Mineral deposits are geologic entities. They differ from other rocks and minerals only in that they contain unusual concentration of some kind or kinds of minerals or metals that have been adapted to use, or they are so located with respect to market and sources of power as to be amenable to low cost extraction. The question of whether or not a given deposit of minerals is of commercial grade and size depends on technology and on all the vagaries of economic and political situations affecting the markets, but the location and character of the deposit itself are fixed by Nature. Moreover, with very few exceptions, mineral deposits can be exploited only once. Consequently, though mining activities come and go within a given district or region, depending on changes in technology, transportation, price, and demand, always the new operations must be addressed to unmined parts of the deposit or to the reworking of mill tailings or waste dumps. This one-crop aspect of the problem means that the local advantages enjoyed by operations in a given area, owing to discovery of high-grade deposits or deposits amenable to production at low unit cost, tend to be short-lived regardless of location. And in view of the rising demand for minerals and metals both at home and abroad, it seems likely, barring political interference, that such local advantages will be even more temporary in the future than in the past.

Trends in Domestic Production Vs Consumption of Key Commodities: The relationships between domestic mine production and U. S. consumption of copper, lead, zinc, and bauxite between 1920 and 1955 are shown in the graphs. In zinc and copper, the production record shows that during those 35 years the country passed from a state of comfortable surplus to one of considerable and growing deficit; in lead and bauxite from near balance to great deficit. Whatever the cause, or the ultimate effect, it seems clear that for these commodities—and among the metals the situations shown are not exceptional—the records of domestic production vs consumption reveal a strong, and growing, state of imbalance.

Reasons for State of Imbalance: Several complicating factors have contributed to this state of imbalance. Most will agree, however, that the dominant reason for this swing is that domestic discoveries of quality sufficient to compete in the world market have not kept pace with rising demands. Known deposits are being used up faster than new ones are being found to take their place. Why has this state of imbalance between discovery and utilization developed?

The problem is not entirely new. It has been developing for several decades, but only in the past 15 to 20 years has it become acute. The trend toward greater utilization and increasing difficulty in discovery of domestic deposits was foreseen a quarter of a century ago, and several far-seeing students of the mineral industries voiced the need for doing something to correct the situation. No one dared, however, to prophesy that the situation



COST OF RESEARCH AS A PERCENT OF SALES BY INDUSTRY

SOURCE: BUREAU OF LABOR STATISTICS AND RESEARCH AND DEVELOPMENT BOARD. AFTER CLYDE WILLIAMS, 1955

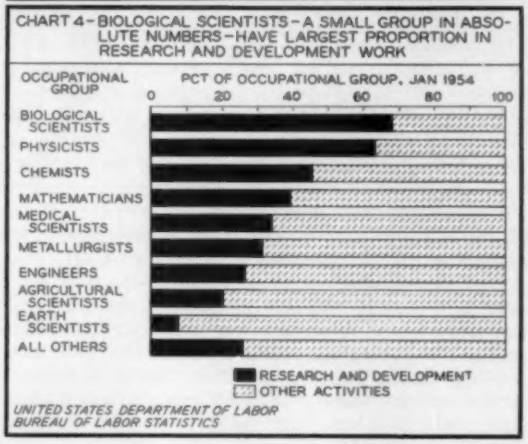


CHART 4 - BIOLOGICAL SCIENTISTS - A SMALL GROUP IN ABSOLUTE NUMBERS - HAVE LARGEST PROPORTION IN RESEARCH AND DEVELOPMENT WORK

UNITED STATES DEPARTMENT OF LABOR BUREAU OF LABOR STATISTICS

would become acute on so many fronts at such a early date. Had these students of the problem been in closer touch with the research laboratories however, they might have been bolder, for hindsight shows clearly that the ravenous appetite and highly complicated tastes of the present-day mineral and metal-using industries grew out of these laboratories, where scientists and engineers created new uses for both old and new minerals and metals, while highly organized sales staffs created widespread demand for their products.

The records also show that the amounts spent in research on new uses and on marketing have been generous, whereas amounts spent in research on discovery techniques have been relatively small. Indeed for a time many of the metal-using industries seemed to have lost sight of the fact that individual deposits of mineral raw materials of given grade and quality are of finite quantity. They seemed also to have overlooked the fact that research on the origin and mode of occurrence of known deposits and the methods of searching for new sources of supply was a necessary element in the industrial process, if capacity to discover new deposits was to keep pace with capacity to utilize those already known. This, the writer believes, is one of the cardinal reasons for the present state of imbalance between the ability to discover mineral deposits and the ability to produce mineral products.

Expenditures for Research: Amounts spent in the U. S. on industrial research over the 25-year period from 1929 to 1954 have increased from roughly $160 million to about $4 billion annually.[1] There is no way of telling how much of this was actually spent on exploration for mineral raw materials, much less the amount spent in research on understanding ore-forming processes and developing new techniques for prospecting. All who have studied the problem agree, however, that the efforts in these fields are and have been relatively small.

The President's Materials Policy Commission tried[2] to estimate the total amounts spent *in research* in the earth sciences (geology and geophysics) through an analysis of the total scientific personnel engaged in basic and development research. Their tentative conclusion was that of an estimated 135,000 scientists and engineers *engaged in research* in the U. S. not more than 3000, or 2.2 pct, are earth scientists (geologists, geochemists, and geophysicists). Assuming this estimate to be correct, and assuming that it cost $15,000 per year to support a scientist, the total annual amount being spent on research in this field would have been $45 million. This is only about 0.3 of the sales value of the raw materials produced in 1951, the year the estimate was made. In view of the fact that by far the greater part of the effort was expended for research in the field of petroleum, the estimated amount left for work on minerals and metals seems small indeed.

Estimates by the U. S. Department of Labor and republished by the National Science Foundation[3] are even more depressing in that they indicate that only 9800, or 1.8 pct, of a total of 553,800 engineers and scientists engaged by industry in the U. S. are earth scientists, and of these not more than 7 pct (or somewhat fewer than 700) were doing *full-time research*.

These two sets of figures, of course, are not directly comparable, since one estimate includes all earth scientists engaged in research and development, and the other only those in industry who engaged in full-time research. Moreover, there is no common definition as to the kinds of work included under the term *research*. Whatever their degree of accuracy, however, both reports support the contention that relatively little is being spent on research in the field of mineral exploration. The records published by the National Science Foundation also show that the amount spent in research on the earth sciences in general is small in comparison with that spent in such sciences as biology, physics, and chemistry.

It seems reasonable to conclude, therefore, that the amount of effort that has been devoted to learning how to find new sources of mineral supply has been small compared with that spent on how to adapt them. This is not surprising when examined in the light of the history of U. S. mineral industries. With minor exceptions — which, except in times of war, could be taken care of easily through imports from low cost foreign sources — the supply problem up to a few years ago has not been pressing. Moreover, in the early days discovery was relatively easy in that the people of this country were privileged to explore and occupy huge and almost virgin territories in which mineral deposits in abundance cropped out at the surface. It called only for courage, imagination, inventive genius, and financial backing to adapt these materials to use and create a demand for them. This was (and is being) done in a way that calls forth unbounded admiration, and

so long as supplies were ample why should great efforts be made to find more? Because of this history, however, is it not reasonable to assume that the basic problem now is not so much lack of deposits amenable to mining as inadequate knowledge of how to find them at a reasonable cost?

Probability Additional Deposits Exist: Most mining districts now active in the U. S. were originally discovered because ore cropped out at the surface, yet the record shows that a large number of the deposits later mined from those districts gave no surface indication of their presence. Instead they were found by careful and persistent probing of the subsurface. This in itself shows that all the ore deposits do not crop out, and is a good reason for believing there are still more to be found.

Ore deposits bear a systematic relationship to the rocks with which they are associated; they usually relate to particular geologic provinces and have formed during specific geologic epochs, moreover statistical records relating to a number of discoveries vs production of ore from typical metalliferous provinces suggest that the amount of ore emplaced at a given time bears a rough relationship to the size of the area affected. A study of mining records of the metal-mining areas in southwestern Utah, southern Nevada, and western Arizona, for example, led Nolan[4] to conclude that the amount of metalliferous ore-bearing material introduced during a given mineralizing epoch bears a rough but constant ratio to the size of the area and that quantitatively the number of commercial ore deposits discovered in the separate mining districts within a given mineral-bearing region is roughly comparable to the size of the area. Hence, other things being equal, it is reasonable to suppose that for depths of at least 1000 ft the number and nature of the deposits occurring beneath the covered portions of a given metalliferous province should be of similar ratio to those exposed at the surface.

It should also be recalled that nearly all the metalliferous deposits that have been found are older, and occur in rocks that are older, than the great blankets of soil, unconsolidated sediments, and many of the lavas and tuffs that cover by far the greater part of the present land surface even in the desert areas of the West. This is to say that the original discoveries in most of the great mining districts in this country were the result of a fortuitous erosion pattern. This particular line of evidence may be negative, but when coupled with the history of discovery and mine production statistics, it too points to the same conclusion, namely, there should be at least as many undiscovered deposits masked by surface cover as have already been found.

Assume, for example, for a region like southern California, where natural erosion uncovered direct evidence of so many of the deposits already found, that the surficial deposits could be stripped off altogether. How many boron deposits would be encountered in the process, and how many additional deposits of the Eagle Mountain, Mountain Pass, and Cerro-Gordo types would then be exposed? Then assume that the bedrock could be sliced off to a modest depth of 1000 ft below the present level. How many new orebodies would then be encountered?

No one knows the answer to these questions, but it seems much more reasonable to assume that many more deposits remain hidden within accessible depths than to assume they are not there simply because we have not yet learned how to find them cheaply and easily.

Earth Sciences in the Search for Commercial Deposits: It is fully apparent that research in mining, mineral dressing, and extractive metallurgy has perhaps turned more rock into ore, in the past 40 years, than the total ore developed by all the geologic, geophysical, and geochemical techniques so far applied. It is assumed, moreover, that engineering and metallurgical techniques must and will continue to play a leading role in ore development. However, despite all the great improvements in mining machinery, mining methods, and beneficiation techniques that have been or are likely to be made, that it will still be necessary to continue to find deposits that contain better than average rock concentrations of ore minerals if a healthy, self-reliant mineral industry is to be maintained. Since these deposits are born of geologic processes it is the responsibility of geologists and geophysicists to devise means for finding them.

The Geologic Sciences in Exploration: The systematic application of geology to problems of ore finding was first introduced to this country in the 1870's by Clarence King and his associates (G. F. Becker, J. D. and Arnold Hague, and S. F. Emmons) during their survey of the 40th Parallel.[5] In the course of their work this unusually able group effectively demonstrated the value of geology as a guide to mineral discovery and mine development. Others contributed, but in large degree the science of mining geology in the U. S. stems from the pioneer work of these men.

The first geological department organized by a mining company was set up in 1898 by Anaconda Co. when H. V. Winchell was engaged to make a systematic geological examination of the company's holdings in the Butte area. The Anaconda geologic staff soon achieved status as an independent exploration unit and has continued to serve with great distinction in that capacity ever since. In due course other companies began attaching geologists to their staffs, and many major contributions to the art and science of ore finding have accrued from the efforts of some of these men. Some companies established departments patterned after Anaconda, but in all too many cases company geologists were attached to the mining departments and employed as samplers and general flunkies rather than as geologists. Thus employed, they had little chance to exploit their training and develop their geological talents. Geologic staffs are a relatively recent adjunct therefore, and the application of scientific methods to the problem of ore finding is still very much in its infancy.

These remarks are by no means intended to disparage the accomplishments of the mining geologists, geophysicists, engineers, and other students of the mineral deposits. Nor are they meant as an alibi or apology for deficiencies in present knowledge of how to search for mineral deposits. They simply emphasize the fact that the scientific approach to prospecting has been little more than tried, and there is much to be learned before proficiency in this phase of the work can be brought up to the proficiency in adaptation and marketing. For more than 100 years an unremitting campaign has been waged to adapt and use minerals and mineral fuel deposits. Except in the petroleum industry, relatively little of the wealth derived from this campaign has been used to develop an understanding of the processes

This article contrasts the great sums spent on research for new uses of materials and on marketing to the small amount spent on research in discovery techniques. Major industrial research installations such as this Westinghouse center near Pittsburgh symbolize the tremendous drive provided many companies and industries through their own programs. In contrast the National Science Foundation estimates that fewer than 700 earth scientists are engaged in full-time research to serve the whole scope of the mineral field.

by which the deposits were concentrated, or to develop the scientific techniques required to guide the search for new sources of supply.

For this reason data on many important aspects of ore-forming processes are far from complete. There are too few data on the physical, electric, radiometric, and thermal properties of rocks, for example, to permit definitive interpretations of many of the geophysical measurements that are possible even now. Despite all the good work that has been done, the nature of ore-bearing solutions is not yet known, nor are the forces that govern their migration understood. Data on the changes in composition that these solutions undergo and on the changes they induce in rocks traversed as they migrate are still far too scanty to warrant definite interpretations. Definitive data are also lacking on the physical, chemical, and biochemical changes that take place in minerals and rocks during weathering, redistribution, and reconsolidation—factors that undoubtedly play a critical role in the development, location, and geometric patterns of many types of ore. For large portions of the country there is not even the comprehensive geologic map coverage needed to determine the geologic framework and establish the geologic sequences of events in the detail required to guide efficient exploration programs.

These are but samples of the kinds of knowledge needed, but not yet available, for use in the search for mineral deposits. Some of these features may seem academic and far-fetched, but geologic and geophysical interpretations, like all other interpretations, are only as good as the data on which they are based. Without comprehensive data on the physical and chemical properties of rock, the meaning of geophysical and geochemical anomalies must remain in the realm of conjecture until they are tested by extensive subsurface exploration. Without more precise understanding of the geochemical nature of ore-bearing solutions, and more definite data on the changes that take place in them and in the minerals with which they come in contact during migration, many subtle differences in the rocks that might serve as guides to ore deposits are likely to go unnoticed.

Present Trends

The picture is not all gloomy. Trends developed in the past few years offer much hope for a bright future in this field. First and perhaps most important is the fact that strong units of the mining industry are rapidly expanding their exploration departments, and some of the larger companies have launched programs of long-range geologic and geophysical research designed to develop new concepts and techniques. This is reminiscent of developments in the petroleum industry in the late 1920's which brought about such salutary results. There is little doubt that had it not been for the aggresive program of research in geology, geophysics, and drilling techniques launched by the petroleum industry and carefully nurtured ever since, the domestic petroleum production record and the petroleum reserve situation would be only a fraction of what they are today. The research efforts of the mining industries to date are relatively small compared to the size of the task ahead, but the trend is in the right direction.

In the recent past, too, there have been successful new or improved techniques developed for use in the direct task of exploration. Among these, geochemical prospecting methods deserve special mention, not only because they promise direct aid in locating mineral deposits but also because they can be expected to yield an immense amount of data on the geochemical behavior of the elements under changing geologic environment — data that are needed to develop better understanding of ore genesis. Geochemical techniques are only beginning to hit their stride. A number of mining companies have taken them up and have trained and equipped their exploration groups to make use of the methods.

Geologic mapping techniques are also being im-

proved in such a way as to obtain greater coverage more rapidly and at less cost. One element contributing to this is photogeology. This method, which blends airphoto interpretation with photogrammetric techniques, is a supplement, not a substitute, for standard ground control methods of geologic mapping, but it cuts down appreciably on the amount of ground control needed to make first-rate geologic maps, and in some cases it gives an overall perspective not readily attained by ground methods.

Perhaps the most exciting trend of the past decade has been the development of motorized precision equipment.[6] Geochemical prospecting in itself is simply a refinement of the old-time panning technique, but the motorized chemical laboratory is new. So are motorized gravimeters, airborne electromagnetometers, scintillometers, portable geiger counters, and a host of other instruments that formerly were laboratory-bound. The overall effect of these developments has been to add a certain amount of precision to the ancient art of prospecting by placing in the hands of the prospectors, exploration engineers, and geologists in the field tools formerly available only to specialized laboratory technicians.

Other new tools and techniques are being made available through recent advances in nuclear science. Isotope ratios of the radioactive elements, for example, have been used for some time to determine the absolute age of minerals and rocks — often a matter of major significance in prospecting. It is entirely conceivable that a study of the isotope ratios of the stable elements may prove of even greater use in unravelling the problems that beset the search for ore. For instance, geologists have known for years that the mineral composition of ore deposits varies systemically within a deposit and from one part of a mineral district to another. These variations, carefully mapped out, have often proved of great practical value in the search for extensions of orebodies or new deposits within a district. Similar systematic variations are beginning to show up in the isotope ratios of common elements found in rocks that have undergone hydrothermal alteration.[7] Work done on certain hydrothermally altered limestones, for example, suggests that the oxygen 18/16 isotope ratio varies appreciably and in direct and systematic relation to the locus of alteration. If it can be established that such relationships are the rule, the study of isotopes may provide a new and highly sensitive guide to hidden ore of the hydrothermal type. The technique is new and relatively untried. Just where it may lead, nobody knows, but it is a wide open and promising field for research.

It seems reasonable to believe, therefore, that the discovery difficulty facing the mineral industries today is not so much lack of mineral deposits to be found as it is lack of knowledge of how to find them at a rate that will meet growing demands and at a cost favorable enough to make their discovery and development competitive in the world markets. One way to alleviate this situation is through aggressive research on the processes by which ore deposits are formed, and the geologic patterns they assume, as well as research to develop better techniques for probing for deposits hidden by surface cover. The success of such a research campaign would, of course, depend upon the imagination, persistence, skill, and astuteness of those who conceive and guide it, and upon the courage and forebearance of those who provide the means to do the work.

An example of what this type of program can do in the case of a material that *becomes* critical in its demand is provided by what happened in the case of niobium. Six years ago, potential demand for this metal far exceeded the potential reserves of all the known sources. Because the element was needed for defense, a boost in price was sponsored by the Government and a study of its mode of occurrence was begun, followed by a search for concentrations amenable to mining, in the face of what then seemed hopeless odds. At the same time a program of metallurgical research was launched in an attempt to modify use patterns so as to conserve this available supply. Both campaigns succeeded in amazing degree. Today it is possible to find and recover more niobium from deposits accessible to U.S. markets than were then thought to exist in the entire Free World. Meanwhile the metallurgists devised ways to obviate the need for the element in one of its major and critical uses.

This case, of course, is not typical, for niobium is a metal that prior to World War II was not of any particular concern, and little attention had been given to its geologic occurrence. Nevertheless, it illustrates what can be done by geological, metallurgical, and managerial talent adequately supported and daringly used.

However, this is not to say that deposits of copper, lead, zinc, and petroleum can be discovered at a rate sufficient to meet current and projected demands. In fact, it is reasonable to expect that in the future the cost of finding and producing some commodities will rise above the economic limit and others will gradually move in to take their place. The limits of some of these transitions can be approximated now—petroleum and gas vs synthetic liquid fuels from coal and oil shales and bauxite vs high alumina clays.

It should be remembered, however, that no mineral commodity has ever gone off the market solely because its source of supply was exhausted. Some have disappeared because research developed other materials to take their places. This is a healthy habit. It is to be hoped that it will continue.

Acknowledgments

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Operation of Liquid Cyclones at Calaveras Cement Co.

Successful installation reduces costs by handling greater tonnage at higher densities

by James T. Curry

FACED with processing an increasing volume of raw material through its existing plant, early in 1955 Calaveras Cement Co. of California considered incorporating liquid cyclones in its raw mill grinding circuits.

One of many problems was the prospect of handling larger volumes of finished slurry through existing pipelines. Fixed capacities of the mechanical classifiers also placed limits on plant expansion.

The expense of duplicating classifiers, pumps, and pipelines could be avoided by installing liquid cyclones to process the larger tonnage of finished slurry at higher densities for pumping through the pipelines already in use. A cyclone installation would replace the bowl classifiers and allow for considerable increase in tonnage at about the same mesh of separation.

A single D-20-B Krebs cyclone, manufactured by Equipment Engineers Inc. of San Francisco, was installed on an experimental basis early in 1955, and although the pump installation feeding the test unit proved to be inadequate, data accumulated on cyclone performance indicated that results demanded by plant operation could be achieved. Orders were placed for four more D-20-B cyclones and four BC Frame Hydroseal pumps, rated to handle the required feed tonnage of 220 tph.

Prior to the installation of cyclones at Calaveras, the raw mill grinding circuit consisted of one 9x9 and one 8x7 Marcy ball mill to reduce the raw cement material to +10 mesh. This was followed by secondary grinding and classification in two 9x25 Marcy tube mills. Bowl classifier product structures are shown in Table I.

Mill feed, consisting of relatively coarse (−2 in.) crushed limestone and shale, imposed a considerable load on the grinding units, and together with the limited capacity of the bowl classifiers, held production to 70 tph per unit. As a logical step to increase production and lower grinding costs, raw material size was reduced by the installation of a 66-in. Telesmith Gyrasphere crusher, operating in open circuit and delivering to raw feed storage a product of −1 in.

The first cyclone installation on a production basis was designed to take advantage of several plant features. Launders and pipelines in place could be used advantageously, and an effort was made to complete the installation with minimum shutdown time for major changes in circuit design. It was important at the time to be able to cut the cyclones in or out of the circuit with as little trouble as possible.

Although the original unit was considered satisfactory from a metallurgical standpoint, operating two cyclones in the same circuit with vastly different feed particle size, dilution, tonnage, and feed pressure requirements left much to be desired. To balance the load between the two cyclones, some secondary pumping was eventually required in bypassing part of the load from one cyclone to the other. Although feed pressure and orifice sizes still differed, better distribution of the load was obtained between one cyclone, operating open circuit, and the other, operating in closed circuit with the 9x25 Marcy regrind mill.

Operating characteristics of the two cyclones comprising A unit are given in Table II, and a comparison of these screen structures with those for bowl classifier operation in Table I reveals a decided increase in −200 mesh material reporting in the cyclone underflow product. As this −200 mesh material is normally the finished product, returning it to the 9x25 mill for further grinding is a wasted effort, although it is sometimes argued that some

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Table I. Bowl Classifier Screen Structures

Mesh	Bowl Feed, Wt Pct Direct	Bowl Feed, Wt Pct Cumulative	Bowl Sands, Wt Pct Direct	Bowl Sands, Wt Pct Cumulative	Bowl Overflow, Wt Pct Direct	Bowl Overflow, Wt Pct Cumulative
14	0.21	0.21	0.32	0.32		
20	1.22	1.43	2.70	3.02		
28	4.97	6.40	8.20	11.22		
35	6.36	12.76	15.12	26.34		
48	6.75	19.51	16.75	43.09		
65	6.10	25.61	15.12	58.21		
100	6.34	31.95	17.02	75.26	0.16	0.16
150	6.93	38.88	13.40	88.66	2.85	3.01
200	4.79	43.67	4.32	92.98	5.15	8.16
−200	56.33	100.11	7.02	100.00	91.84	100.00
	114 tph		71.5 pct solids		15.0 pct solids	
			44.5 tph		70 tph	

Operating Results:

Tables I and II permit comparison of bowl classifier and cyclone operation, while Tables II and III permit comparison of first and second cyclone installations. There were practical limits in how well the cyclones could produce underflow free from −200 mesh material.

Table II. Cyclone Data for Unit A

Cyclone No. 1. Feed inlet pressure, 15 psi; vortex, 6 in.; apex, 3⅝ in.

Mesh	Feed, Wt Pct (Calculated) Direct	Feed, Wt Pct (Calculated) Cumulative	Overflow, Wt Pct Direct	Overflow, Wt Pct Cumulative	Underflow, Wt Pct Direct	Underflow, Wt Pct Cumulative
−10 +20	2.81	2.81			3.91	3.91
28	5.48	8.29			7.62	11.53
35	8.92	17.21			12.40	23.93
48	10.91	28.12			15.19	39.12
65	9.86	37.98			13.70	52.82
100	9.67	47.65	Trace	Trace	13.45	66.27
150	8.39	56.04	1.65	1.65	11.02	77.29
200	5.31	61.35	5.90	7.55	5.08	82.37
−200	38.65	100.00	92.45	100.00	17.63	100.00
Total	100.00		100.00		100.00	
	55.5 pct solids		33.7 pct solids		76.8 pct solids	
	637 gpm		357 gpm		262 gpm	
	135.7 tph		38.2 tph		97.5 tph	

Cyclone No. 2. Feed inlet pressure, 30 psi; vortex, 5¼ in.; apex, 2⅝ in.

Mesh	Feed, Wt Pct (Calculated) Direct	Feed, Wt Pct (Calculated) Cumulative	Overflow, Wt Pct Direct	Overflow, Wt Pct Cumulative	Underflow, Wt Pct Direct	Underflow, Wt Pct Cumulative
−28 +35	0.41	0.41			0.76	0.76
48	4.02	4.43			7.45	8.21
65	5.45	9.88			10.10	18.31
100	11.58	21.46	0.30	0.30	21.20	39.51
150	15.31	36.77	5.54	5.84	23.46	63.15
200	10.72	47.49	9.70	15.54	11.61	74.76
−200	52.51	100.00	84.46	100.00	25.24	100.00
Total	100.00		100.00		100.00	
	48.7 pct solids		34.4 pct solids		75.5 pct solids	
	737.5 gpm		542 gpm		195.5 gpm	
	129.7 tph		59.6 tph		70.1 tph	

Mesh	Composite Overflow to Thickeners, Wt Pct Direct	Composite Overflow to Thickeners, Wt Pct Cumulative	Composite Underflow to Regrind, Wt Pct Direct	Composite Underflow to Regrind, Wt Pct Cumulative
−10 +65			38.40	38.40
100	0.18	0.18	16.69	55.09
150	4.02	4.20	16.30	71.39
200	8.21	12.41	7.81	79.20
−200	87.50	100.00	20.80	100.00
Total	100.00			
	34.1 pct solids		76.1 pct solids	
	899 gpm		457.5 gpm	
	97.8 tph		167.6 tph	

Table III. Combined Results of Cyclone Data for Unit B

Feed inlet pressure, 11 psi; vortex, 5¼ in.; apex, 2⅝ in.

Mesh	Feed, Wt Pct (Calculated) Direct	Feed, Wt Pct (Calculated) Cumulative	Overflow, Wt Pct Direct	Overflow, Wt Pct Cumulative	Underflow, Wt Pct Direct	Underflow, Wt Pct Cumulative
10	0.08	0.08			0.18	0.18
14	0.16	0.24			0.38	0.56
20	0.65	0.89			1.52	2.08
28	1.67	2.56			3.90	5.98
35	3.70	6.26			8.64	14.62
48	4.39	10.65			10.23	24.85
65	5.45	16.10			12.67	37.52
100	7.14	23.24	Trace	Trace	16.64	54.16
150	8.36	31.60	2.14	2.14	16.64	70.80
200	8.96	40.56	9.42	11.56	8.35	79.15
−200	59.44	100.00	88.44	100.00	20.85	100.00
	45.6 pct solids		34.0 pct solids		71.0 pct solids	
	932 gpm		728 gpm		204 gpm	
	152.3 tph		86.9 tph		65.4 tph	

Table IV. Comparative Performance Data

Item	Mechanical Classification	A Unit Cyclone Classification	B Unit Cyclone Classification
Tons per hour, feed	70	100	100
Slurry to thickeners, percent solids	13.5	35.0	35.0
Ratio, solids:water	1:6.4	1:1.9	1:1.9
Pulp volume per ton of solids, cu ft	216.9	71.4	71.4
Total slurry, gpm	1900	892	892
Feed increase, pct		43.0	43.0
Decrease in gallons per minute, pct		53.0	53.0
Total pumping load, hp	117	126	84
Total kilowatt-hours per ton of feed	1.213	0.939	0.627
Power decrease, pct		22.6	48.4

−200 mesh material promotes better regrinding by providing a more viscous mass inside the mill.

It appears that one of the inherent disadvantages of closed circuit cyclone classification is the relative inefficiency, as compared to bowl classification, in making a clean underflow separation of the fine or finished material. The problem can be remedied somewhat by further dilution water or by increased feed entrance pressure, neither of which is practical or desirable from an operating standpoint. The entrance pressure of 30 psi at No. 2 cyclone already exceeded the limits of economic pump operation, and further increase in feed dilution would defeat the purpose of providing a high density overflow product.

On the basis of data accumulated for six months on the A unit performance, it was decided to install the new B unit in parallel to relieve the problem of excessive fines in the underflow. The necessary changes in plant design, incorporated when the 9x9 Marcy ball was installed, have obtained the following benefits:

1) Equal pump operation, wear characteristics, and power requirements.

2) Even distribution of load and particle size between the two cyclones.

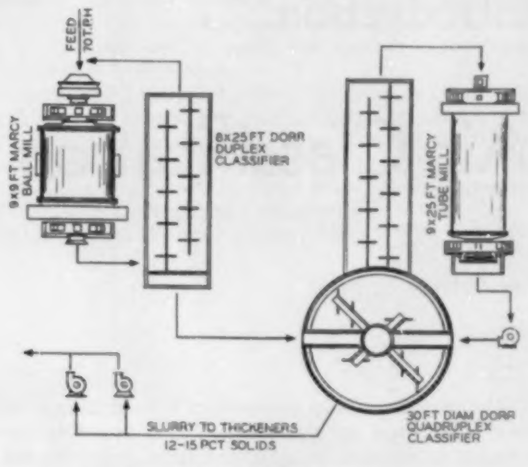
3) Equal cyclone operation, feed pressure requirements, and wear characteristics.

4) Equal balance and distribution of cyclone products to thickeners and regrind.

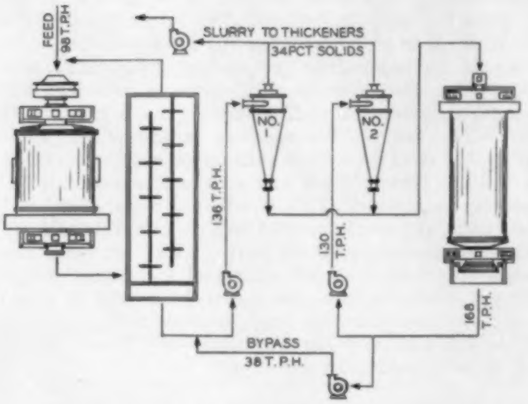
5) Standardization of wearing parts, such as apex valves and vortex orifices.

Flowsheets:

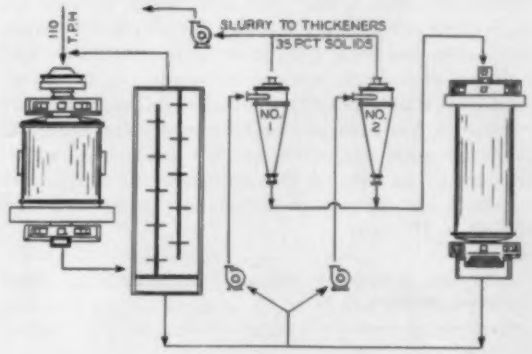
Starting point for development program was conventional flowsheet at right. The "A" cyclone circuit was designed to take advantage of existing plant features and minimize time for installation. The "B" circuit, installed to minimize fines in the underflow, permitted balancing the load and other factors. Cyclone installation resulted in reduction of 4.6 tons of water handled per ton of solids produced, with attendant power saving, and enabled plant to handle 43 pct more material at a reduction of 53 pct in gallons of slurry pumped per minute.



Conventional flowsheet



"A" Cyclone flowsheet



"B" Cyclone Flowsheet

Although Tables II and III show much the same screen structures for overflow and underflow products, these results are not directly comparable because feed rates to the B unit cyclones were somewhat below normal during the sampling period represented by Table III. The underflow solids of 71 pct reflect a low weight loading factor, resulting in a higher percentage of −200 mesh material than is normally the case.

The apex valves, of molded natural rubber, last about 100 days and are the only parts of the cyclone that show appreciable wear. Several types of hydraulic and pneumatic controls have been installed to compensate for this wear and to maintain a fixed orifice diameter. Of these, the pneumatic type of valve has successfully maintained the fixed orifice. It remains to be seen whether it will provide a longer life for the rubber valve.

Another innovation that shows promise is the apex valve with ceramic insert. It is hoped that this extremely hard, high-temperature, superceramic material will maintain a fixed diameter for a long period, but it is far too early to make predictions.

Because large quantities of water were originally required for the bowl classifiers to effect the desired separation at 88 pct −200 mesh, it was necessary to handle a finished slurry at 12 to 15 pct solids —a solids to water ratio of about 1:6.5. Transporting this volume to the thickeners imposed a severe load on pumps, pump motors, and pipelines that had been installed at an earlier date when production was substantially lower. With a further anticipated increase of 35 pct in production the only alternative to duplicating equipment was to reduce the volume of water to be pumped by producing slurry at a higher density.

This was attained by using the liquid cyclones to produce a finished ground material, within the accepted limits of size distribution, at 35 pct solids, or a solids to water ratio of 1:1.9, eliminating 4.6 tons of water for each ton of solid material handled. A gain in economy is achieved not only in pumping slurry to the thickener but also in handling less thickener overflow water back to the plant process. About 43 pct more material is now being successfully processed at a reduction of 53 pct in gallons of slurry pumped per minute. The increased tonnage is being handled with a power saving 23 to 48 pct.

It is only natural that a lot of lost motion occurs, in the form of trial and error, when radically different types of milling equipment are installed in an operating mill circuit. Calaveras Cement Co. anticipates that with improved operating techniques there will be even greater savings per ton of finished material.

Introduction

Air Classification Symposium

by Arthur L. Hall

The purpose of this symposium is to challenge industry to better air classification, and as a first step toward fulfillment, to spotlight three fairly recent, aerodynamically designed, highly efficient air classifiers. This discussion does not intend to imply that present practice is bad. Actually, when present air separators are used for the jobs for which they were originally designed, they give high production and uniform operation, with a minimum of maintenance. The customers of nonmetallic mineral pigment producers, however, are making increasing demands for finer products, such as 99.9 pct -20μ, which appears to be beyond the economic capabilities of standard air separators industrially used over the last 10 to 30 years.

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Here is an actual example. A 10-ft diam pebble mill is in closed circuit with a conventional 12-ft diam centrifugal air separator, grinding 20-mesh wollastonite to a series of fine grades up to 99.97 pct − U.S. No. 325 mesh (44μ). Fig. 1 shows variations in fineness of mill discharge (or air separator feed), recycle for regrinding (or air separator tails), and product as measured by a wet rotating sieve test. For example, when the mill is turning out a product 80 pct −325, the mill discharge or separator feed is 30 pct −325, and the coarse tailings or recycle is only 5 pct −325. Hence 25 pct of the 30 pct available is accepted as product. This is efficient separation, and the ratio of recycle to feed is only about 2:1. However, to make a product testing 99.97 pct −325 the separator feed is 62 pct −325 and the coarse tailing 60 pct −325. In this case the recycle ratio is about 13:1 and the economic limit to produce fineness has been substantially reached with this given equipment and process arrangement.

Fig. 1

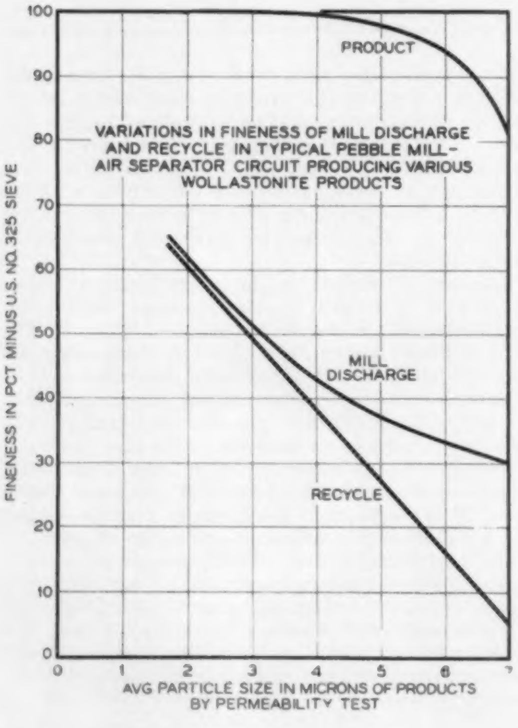
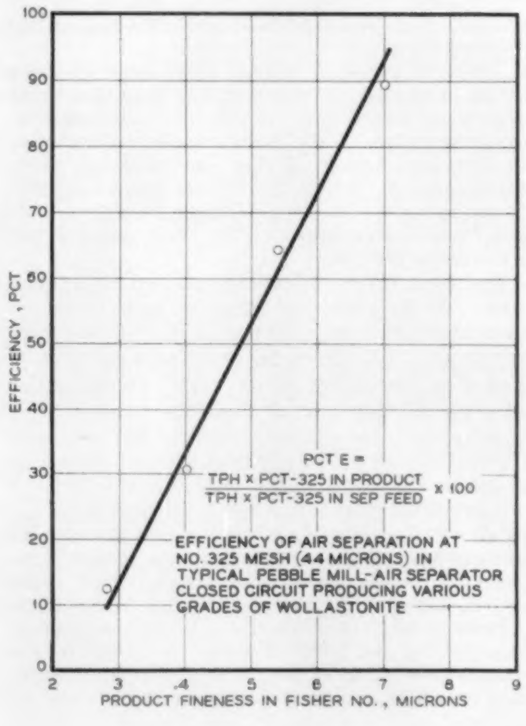


Fig. 2



This last example points out that the pebble mill is doing a better job of grinding than the separator is of classifying, and this is the primary reason why a symposium on classification and not on milling was arranged.

In any statement of efficiency, the exact formula and particle size *cut-point* must be clearly defined. Efficiency of air separation may be expressed as the quantity of material finer than the desired cut-point which is obtained in the product, expressed as a percentage of the quantity of material finer than this cut-point which was present in the classified feed. This is stated in equation form in Fig. 2., the *A* micron size being the desired cut-point, in this case, 44μ. From this data it is apparent that production of a $3\text{-}\mu$ product on this particular system is far from theoretical perfection.

Efficiency is a good guide but does not in itself solve the problem. The above discussion should not overshadow the fact that the pigment producer wants equipment requiring minimum capital investment, minimum operating and maintenance expense, at the same time giving a maximum production rate, while making the superfine products now in demand.

The Optimal Vortex Classifier

by Robert E. Payne

UNTIL recently classifier design was a mysterious art consisting of cut-and-try experiments to improve some existing winnowing machine. Powder was dumped into an airstream of very non-uniform cross section with little or no effort to control initial velocity, secondary flow patterns were present to keep remixing the powder, and classifying efficiencies were very low. If a particle got off to a bad start or was caught in a secondary flowstream it was misclassified, and only a second pass could rectify the error.

About ten years ago a new concept began to appear—the *constant cut-point* classification zone. Suppose that it were possible to retain a particle in the classification zone until the effects of initial velocity disappeared, and further, that no matter where this particle found itself at that time, it would be forced in a given direction if it were less than a certain size and in the opposite direction if it were greater than that size. Then even if the particle were to start out in the wrong direction, it would eventually turn around and go the right way. One requirement in creating such a constant cut-point region was the suppression of secondary flow patterns. Out of these considerations, the idea of a *controlled vortex* classifier began to appear. Machines of this type were actually produced commercially by Alpine in Germany, Bahco in Sweden, and a few other companies. Sharples Corp. built several comparable experimental machines that were tested commercially by other companies, but it was apparent that the *controlled vortex* classifiers, in spite of their superiority over earlier *winnowing* machines, were disappointingly far from the ideal. In general, the early winnowing classifiers gave 10 to 40 pct product recoveries.* The controlled vortex classifiers gave product recoveries from 20 to 50 pct, depending on the material being classified, the cut-point of classification, and the feed rate. Sharples Corp. decided to delay production of a commercial classifier until the missing elements in the controlled vortex classifier could be found.

* *Product recovery* as used here is usually referred to as *efficiency* by people working with classification equipment. It is the percent of available product material in the feed that is removed as product by a single pass through a classifier. If 80 pct by weight of the feed material is composed of particles smaller than 20μ and 50 pct of that amount (or 40 pct of the feed material) is delivered as product, the product recovery is 50 pct. This definition gives generally higher numbers than the true efficiency, since it makes no mention of the percent by weight of product which is greater than the 20μ separation point. This has been well treated by H. W. Newton and W. H. Newton in "A Study of Classification Calculations," *Rock Products*, Aug. 13, 1932, p. 26.

Extensive measurements with several models, as well as detailed mathematical investigation of particle trajectories in vortex classifiers, revealed that another important development was necessary before a highly efficient controlled vortex classifier could become a reality. While in principle a controlled vortex classifier could be adjusted to create a constant cut-point classifying region for any feed material of a given particle size distribution and a certain feed rate, any change in either the particle size distribution of the feed material or the feed rate would introduce an interaction with the spiraling airflow which would immediately destroy the carefully adjusted constant cut-point region, with a consequent drop in classifying efficiency and sharpness of cut. An immediate solution to the problem might appear to be simply to maintain a high ratio of mass airflow to powder so that the interaction of

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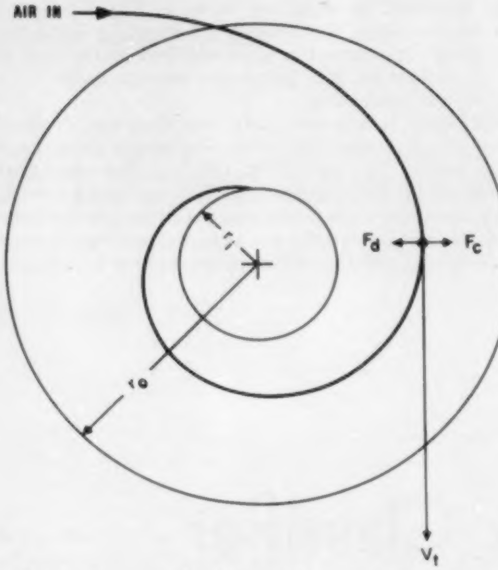


Fig. 1—F_c = centrifugal force; F_d = drag force; V_t = tangential velocity; r_o = outside radius, classification zone; r_i = inside radius, classification zone.

the powder on the air would be small. This is possible for small pilot plant classifiers but is completely out of the question because of blower costs and power costs and power requirements in the range of commercial classifier capacities. This situation is probably the true basis for the remark that anyone can build a good laboratory or small pilot plant classifier but that these units can never be scaled up without drastic loss in efficiency.

There is fortunately one (and only one) way to circumvent this problem. *There is one particular mathematical spiral vortex that can be maintained completely independent of particle size distribution in the feed material and feed rate.* The Sharples super classifier is based on the incorporation of this particular spiral vortex into a constant cut-point classifying zone. It has been designed to minimize blower capacity and power requirements and can be built in any size with *identical classification performance.* Several units have been built and tested in the capacity range from 200 lb per hr to 20 tph. Larger units are being considered.

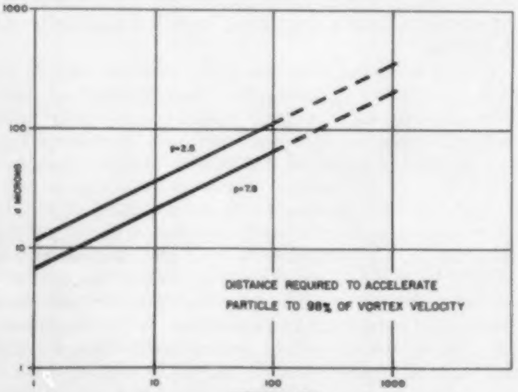
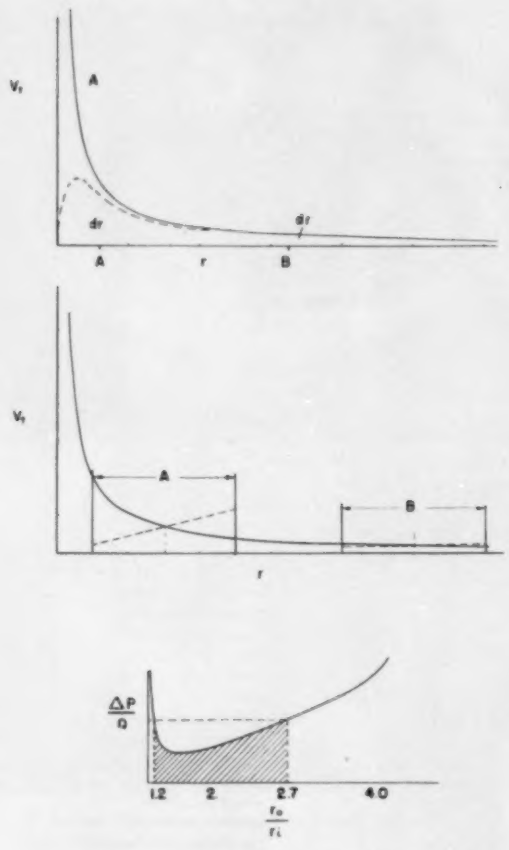


Fig. 2—Plot shows that 100-μ particle of talc must travel 80 ft before approaching air stream velocity—a distance greater than the trajectory available in the classifier.

General Characteristics of Vortex Classifiers: Fig. 1 illustrates the general nature of any vortex classifier. The classifying zone is that volume between the cylindrical limits of radii r_o and r_i. Air introduced at the outside of the classifier is directed by vanes into a spiral flow. A particle in the classifier region will have a radially outward centrifugal force F_c acting on it because of its tangential velocity V_t and a radially inward drag force F_d acting on it caused by the inwardly spiraling air. These forces can be made to balance for any single particle size at any point in the classifying zone if *proper airflow can be maintained at every point in the classifying region and particles can be brought up to air speed before they leave the classifying zone.* As an example of this, in one type of controlled vortex classifier an attempt is made to maintain an Archimedian spiral in the classifying zone, i.e., the tangential velocity of the air is kept constant throughout the classifying zone. This spiral will produce a constant cut-point classifying region for a parallel plate classifier. Now this can be accomplished only if energy is extracted from the air at a very definite rate as it moves inwardly. This energy must not be removed by the walls of the classifying zone or else secondary flow patterns will be created and the uniformity of airflow in the axial direction will suffer badly; consequently, the energy must be extracted by friction losses of the vortex on itself and through energy transferred to the powder being classified by the air in the form of tangential acceleration of the particles as they move radially inward. This condition can be approximated at a single value of feed rate and for a single particle size distributon of the feed powder as long as the total mass of powder involved is small compared to the total mass of air; consequently, fairly efficient classifiers can be built on this principle in small sizes. As already mentioned, classifiers of this type that are capable of making a fine product at *high classification efficiency* at capacities exceeding a few hundred pounds per hour have prohibitive blower requirements. On the other hand, in making a coarse product the constant cut-point region may be largely lost unless special methods are used for feeding. The larger particles cannot be accelerated to the tangential speed of the air by the air itself in the time they are in the classifying region. This can be seen from Fig. 2, e.g., a 100-μ particle of talc must travel about 80 ft to be accelerated by the air stream to 98 pct of the air stream velocity. This is many times greater than the length of the particle trajectory in the classifying region.

A great many types of spiral flow other than the Archimedean spiral can be used in vortex classifiers if the rotational energy of the air can be changed by the powder in the proper way. With the single exception of *free vortex* flow, which is used in the Sharples classifier, all other spirals have the same disadvantages as the Archimedean spiral. There is a basic reason for this. The *free vortex* is the only vortex in which the angular momentum of the air is conserved. If no forces are allowed to act upon the airstream in the tangential direction, it will conserve its angular momentum and will have a free vortex flow. If no tangential forces are exerted on the particle while it is in the classifying region it will also conserve its angular momentum; consequently, if a particle of any size is introduced into the classifying zone at any point with the same angular momentum as the air, *the tangential velocities of the air and powder will remain perfectly matched at every point*



Figs. 3a (top), 3b (center), 3c (bottom). Refer to text.

in the classifying region and there will be no interaction between the powder and the air to alter the free vortex. In the free vortex classifier, the constant cut-point region can be maintained completely independent of feed rate and particle size distribution changes. Further, it is possible to single out the design of a free vortex classifier which will meet given performance standards with a minimum of blower capacity. For perfect operation, the blower should be of the constant volume type, but centrifugal blowers can also be used where feed rate does not vary excessively.

Design of the Free Vortex Classifier: Since a free vortex is to be maintained in the classifying region, it will not be necessary to depend on the air to accelerate the particles to their proper speed. These particles will be accelerated mechanically to the speed of the vortex and introduced into the vortex with the same tangential velocity the air has. Now the powder will not influence the vortex flow, and the problem discussed with reference to Fig. 2 is eliminated. Fig. 3a shows how the tangential velocity in an actual vortex (dotted line) compares with that of a free vortex because of internal friction due to air viscosity. The self-induced sheer forces are very high at *A* and relatively unimportant at *B*; consequently, the classifying region should have a doughnut shape and the inner radius should be well removed from the center. For minimum secondary flow and friction losses due to the side walls, these should rotate with the mean velocity of the air.

$$\alpha_c = \frac{18\eta}{\rho d_c^2} C_s$$

C_s = Stokes drag coefficient
d_c = cut-point diameter
ρ = particle specific gravity
ρ_a = density of the air
η = viscosity of air (or fluid)
r = radial coordinate
V_r = radial velocity of the air
V_t = tangential velocity of the air and also of the powder
m = mass of the particle
V_{t_o}, r_o, etc. = value of quantity at outside edge of classifying zone
V_{t_i}, r_i, etc. = value of quantity at inside edge of classifying zone
Q = volume velocity of air
$h(r)$ = spacing of walls at radius r
Δp = pressure drop from r_o to r_i due to vortex flow

These plates are therefore ideal for accelerating the powder to the speed of the air. In Fig. 3b, it can be seen that the differential velocities between the vortex and the rotating walls are large in region *A* and negligible in region *B*, so that again a doughnut-shaped classifying region well removed from the center is required if secondary flow patterns are to be avoided. All tangential forces acting on the air have now been eliminated so that the vortex cannot be other than the desired free vortex.

The classifying region is to have a constant cut-point. Consequently, if d_c is the particle diameter at the cut-point, the centrifugal force of the particle is equated to the radially inward drag force

$$\frac{mV_t^2}{r} = C_s 3\pi \eta d_c V_r \text{ or } V_t^2 = \alpha_c r V_r \qquad [1]$$

$$\alpha_c = \frac{18\eta}{\rho d_c^2} C_s$$

Since the particle conserves angular momentum,

$$V_t = V_{t_o} \frac{r_o}{r} \qquad [2]$$

and in terms of volume air flow, Q

$$V_r = \frac{Q}{2\pi r h(r)} \qquad [3]$$

when Eq. 2 and Eq. 3 are substituted into Eq. 1

$$V_{t_o}^2 \frac{r_o}{r^2} = \alpha_c \times \frac{Q}{2\pi\, h(r)}$$

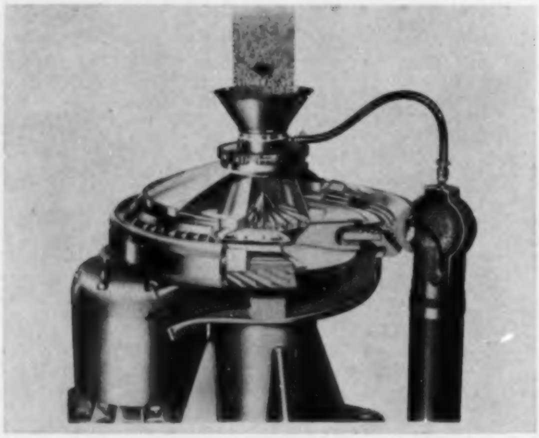
or

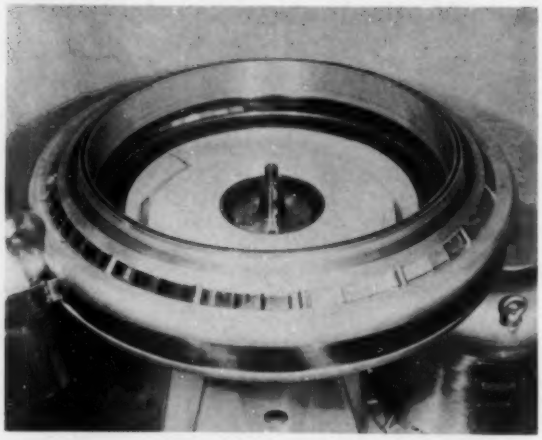
$$h(r) \propto_c r^2$$

that is, the walls should be paraboloids.* Over the restricted range of interest in a doughnut-shaped classifying region, this shape may be made conical with no significant error and in fact with several small advantages.

* This is not exactly true, since α_c contains the factor C_s, which depends upon r. This correction is unimportant for Reynolds numbers below about 5 but should be included when larger Reynolds numbers are considered. In the Sharples classifier, it amounts to approximately a 20 pct correction factor and may be compensated for by increasing the plate spacing very slightly.

Finally, consider the pressure drop due to the vortex in the classifier. Actual flow losses are small, since the total air flow is very nearly in a tangential direction and the tangential wall losses have been





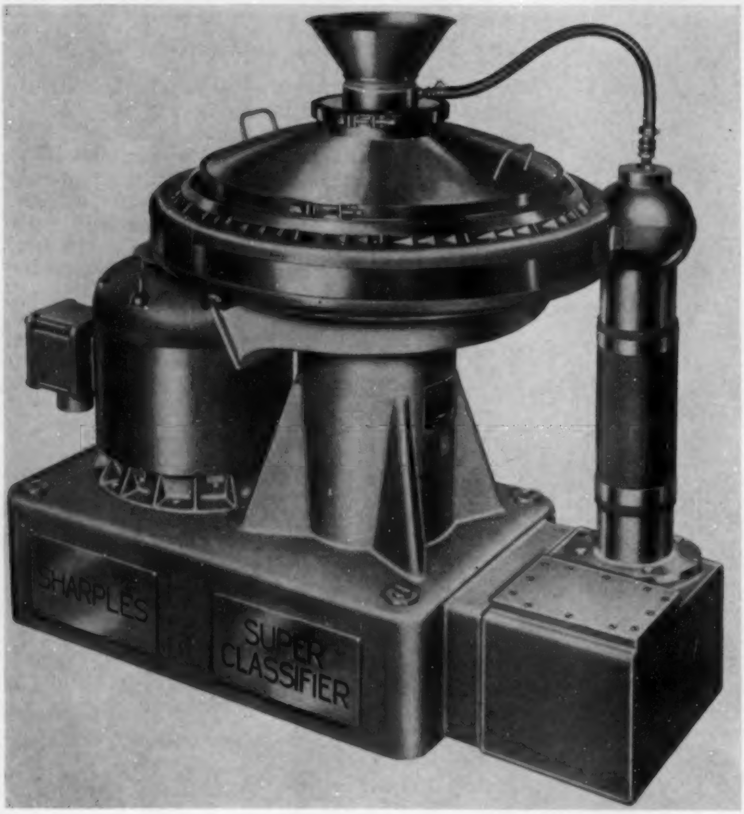


Fig. 4 (upper left)—After powders are passed through the super classifier's separation zone, the fines fraction is carried by the exhauster air stream to a collector; the coarse fraction is carried out through the air lock of the classifier, from which it is either returned to the mill for regrinding or collected as a product.

Fig. 5 (above)—Close-up of the classifier with replaceable insert parts removed. Fig. 6 (lower left)—The Sharples super classifier is unusually small and compact. The 5-tph model occupies about the same space as an office desk.

eliminated. The only significant pressure drop is that due to the vortex itself and is simply

$$\Delta p = \frac{1}{2}\rho_a V_{t_i}^2 = \frac{1}{2}\rho_a V_{t_o}^2\left(\frac{r_o}{r_i}\right)^2$$

Now an important figure of merit of a classifier of a given capacity capable of achieving a given cut-point is the pressure drop necessary to achieve the required volume velocity of air, since this specifies the size and horsepower requirements of the blower or exhauster. In the free vortex classifier this is

$$\frac{\Delta p}{Q} = \frac{\frac{1}{2}\rho_a V_{t_o}^2\left(\frac{r_o}{r_i}\right)^2}{2\pi r_o h_o V_{r_o}} = \frac{k}{r_o} f\left(\frac{r_o}{r_i}\right)$$

If $\frac{r_o}{r_i}$ is chosen properly $\frac{\Delta p}{Q}$ will be a minimum so that the smallest possible blower can be used. This occurs when $\frac{r_o}{r_i} = 1.5$, which agrees perfectly with the doughnut-shaped region well removed from the center. The curve of $\frac{\Delta p}{Q}$ is shown in Fig. 3c. This completes the theoretical design of the free vortex classifier.

The mechanical design is shown in Fig. 4. Feed powder is introduced into the middle of the classifying zone from the top rotating plate or wall. Particles smaller than the cut-point are carried inward with the air and then down and outward again so

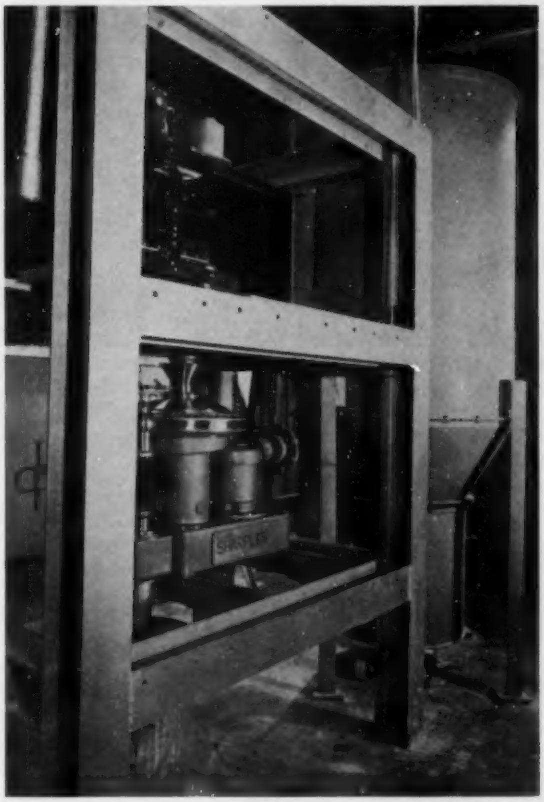


Fig. 7—Part of the large pilot plant, in which a wide range of materials has been tested.

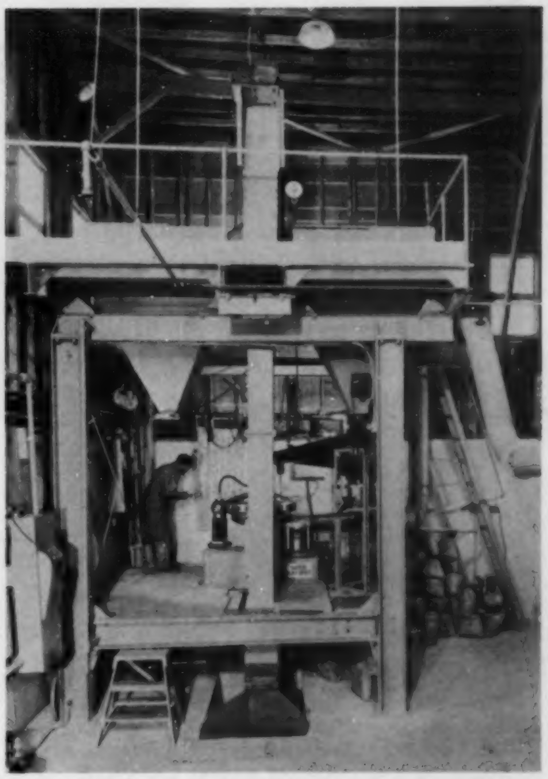


Fig. 8—Operating in typical powder processing plant, this 5-tph Sharples super classifier is shown with associated equipment.

that a significant amount of vortex pressure drop is recovered at the discharge volute. Coarse particles are thrown outward by centrifugal force and are removed by a series of coarse discharge rings leading to a tangential takeoff slot. The air is brought in independently from the powder by the vane system shown separately in Fig. 5. Fig. 6 is a photograph of a 16-in. production classifier of 1-tph feed capacity. The overall height is approximately 33 in.

Last, but far from least, are the test results on the free vortex classifier. After a preliminary check of the theory on a small experimental unit capable of about 300 lb per hr, the unit was scaled up to 15-tph capacity representing a scale-up factor of about 1 to 100. The large machine performed exactly like the small one, again verifying the theory. The present

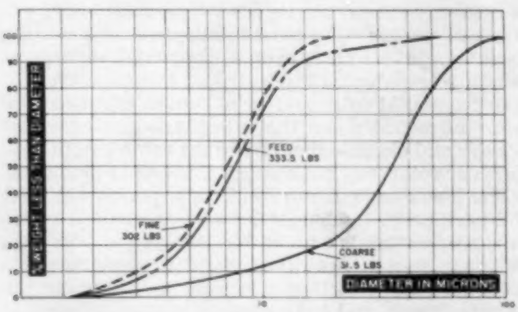


Fig. 9—Particle size distribution analysis for classification of a talc product.

production line consists of two sizes of machines having rotor sizes of 16 and 27 in., and feed handling capacities of 1 and 5 tph when producing a fine talc or limestone product with a 325 mesh screen residue of less than 0.02 pct. A 34-in. unit of 10-tph capacity will soon be available. Product recoveries in the 80 to 90 pct range are normal, some applications approaching 100 pct. Fig. 9 shows a classification very close to perfection on a talc product. Efficiency in this case was 96.3 pct. Of about 310 lb of available *fine* material in the feed, only 8 lb were lost to the coarse fraction. The feed rate of this particular classification was 2 tph on a 5-ton machine.

Although efficient classification of material on the basis of $d\sqrt{\rho}$ (particle size x $\sqrt{\text{specific gravity}}$) is now a reality, it should be recognized that practical problems still exist for many individual process engineers. Sometimes the separation desired does not depend strictly on $d\sqrt{\rho}$, in which case a correlation must be made between the required property and $d\sqrt{\rho}$. This is a characteristic common to all air classifiers. Problems of abrasion, contamination, chemical activity, and possible product degradation must also be considered. To aid the industry in this respect, Sharples Corp. has established two pilot plants where particular classification problems can be undertaken on a large scale. Fig. 8 shows a part of the smaller pilot plant in which tests can be made at any feed rate up to about 1 tph. Fig. 7 shows part of the large pilot plant in which tests at 10 tph can be made. A wide range of different materials has been studied in these installations.

Grinding and Classifying in the Subsieve Range

Latest development in classifying and grinding, the Lykken open rotor, fluid energy mill is adaptable to a wide range of applications.

by William H. Lykken

REDUCING any material to a specified particle size in the subsieve range (below 200 mesh) presents many problems. Above the subsieve range, classification is a matter of sieving and screening, a comparatively simple process. Because particles in the grind have more body, they are more susceptible to further reduction by crushing or impact. In the subsieve range, a large amount of material must be reduced and classified to all -20 to 10 or 5μ, with no oversize tolerated.

1 in. = $25{,}400\mu$.

1 cu in. = $24{,}400^3$ micron cubes.

1 lb of material of unit (one) specific gravity = 27.7 cu in. of solids.

Hence, for every pound of material reduced to micron cubes:

$$\frac{25{,}400^3 \times 27.7}{\text{Sp. Gr.}} = \frac{454{,}000{,}000{,}000{,}000}{\text{Sp. Gr.}} \text{ particles.}$$

To obtain the number of particles of any micron dimension in a pound of material, divide the above figure by the cube of that dimension. Then, since most material reduces to rounded particles, that is, cubes with the corners knocked off, multiply the result by X.

These figures constitute the problem, both in reduction and classification. *En masse*, they represent a fluid, and the product must be classified as a fluid, highly and uniformly diluted in air. The only known method of classifying dry subsieve particles is to allow the desired particle size and finer to follow a suspending air flow, while at the same time restraining the oversize from doing so.

Obviously this calls for a high degree of dilution. There must be adequate space between the particles so that each particle can move freely within the mass and through it. This calls for at least one pound or 13 cu ft of air per pound of solids.

W. H. LYKKEN is Sales Manager, Hurricane Pulverizer Co., Minneapolis.

Adequate dilution is therefore the first factor, requiring a classifier with comparable and adequate volumetric capacity.

Complete and uniform distribution of the material in the air as individual particles is the second essential, but equally important is the fact that this condition must be continuously maintained in the classifier.

Complete aeration is the third essential. Material entering the classifier must be thoroughly disintegrated and aerated to insure that every particle is surrounded with its individual film of air, and that condition must be maintained in the classifier.

This is generally overlooked, so that two or more particles agglomerate, owing to static or other causes. Only an air film can individualize a particle and lubricate its movements.

Principle of classification can, under the above conditions, be based on the drag of an airflow on particles suspended in the flow, which varies with the first power of their diameter, and centrifugal force, which varies as the cube of their diameter, in the opposite direction.

The Lykken open rotor, fluid energy mill was designed with a classifier for subsieve classification, involving the considerations, requirements, and principles outlined above. The open rotor, fluid energy mill consists of a horizontal cylindrical housing mounted, in this case, over a horizontal open rotor grinding or reducing mill.

A peripheral inlet and axial outlet, multipurpose rotor is mounted in the classifier housing. Both grinder and classifier rotors are the same length. The classifier rotor may have an axial outlet at both ends.

The lower or mill rotor acts as a peripheral inlet and peripheral discharge fan and provides a recirculating flow all its own within the system. The fines and near fines are continuously discharged tangentially into the classifier chamber, up and over the classifier rotor, and back down into the discharge of the lower rotor. Any fines in the returns recirculate above the reducing rotor. Any oversize goes back

to the grinding rotor for further reduction. Note that this is a constant, self-contained, individual figure eight circulation, induced by the lower rotor as a peripheral fan and by the ejector effect at the top of the rotor.

The material is fed in controlled amounts for the full length of the lower rotor. All the air is supplied tangentially for the full length of the classified rotor at the top of the classifier housing. Airflow through the system is provided by an *independently* driven exhaust fan associated with the collector system.

The classifier rotor is constructed with a peripheral ring of spaced rods and an inner axial drum consisting of spaced annular disks. This annular disk drum provides a laminar inlet port, in depth, to the axial outlet duct. The row of spaced rods surrounding this disk drum produces a classifying effect by an induced vortex action developed within and between each rod.

This rotor has many functions:

1) Maintaining a continuously circulating load to be classified in the classifier housing.

2) Providing the desired contrifugal stresses in the rotating mass proportional to its rotative speed.

3) Serving as a pre-classifier because its spaced rods continuously and selectively suck in all the fines in the material circulating in the classifier housing, allowing oversize to return to the mill.

4) Maintaining a uniform distribution of material the full length of the rotor.

5) Maintaining a continuous state of aeration or particle air film effect.

6) Maintaining a uniform particle distribution in the air.

7) Serving as an air distributor and air mixer.

8) Serving as an essential element in the final classification effect.

It has been noted that all circulating fines are continuously sucked into the rotor. This in-and-out flow, or continuous sorting, is a function of the rotor itself and independent of any airflow through the classifier.

The particles enter the rotor at a distance inversely proportional to particle size. There are no large particles. There is colloidal dust only to the point of maximum penetration. This is a matter of rod spacing and rotating speed, by design.

With no air flowing through the classifier, this point of maximum penetration may fall far short of reaching the axial outlet, if the rotor is so designed. The classifier rotor, in the area between its rod circle and its axial drum, merely rotates with a core in depth of inwardly classified material—continuously present, continuously renewed.

When air is drawn through the rotor, it enters the periphery with the recirculating airflow as additional and scavanger air. It will be observed that it enters and expands the loops of the circulating flow in the rotor.

At low rate of flow, it extracts only the inner layers of the circulating flow—the superfine, and always *all* the superfine. Then, as the flow is increased, increasingly larger particles are withdrawn proportional to the rate of flow.

In this set-up it is not necessary to capture a given particle size the first time it gets within range of the outlet. All particles finer than the specified top size go in on the first approach, keeping the

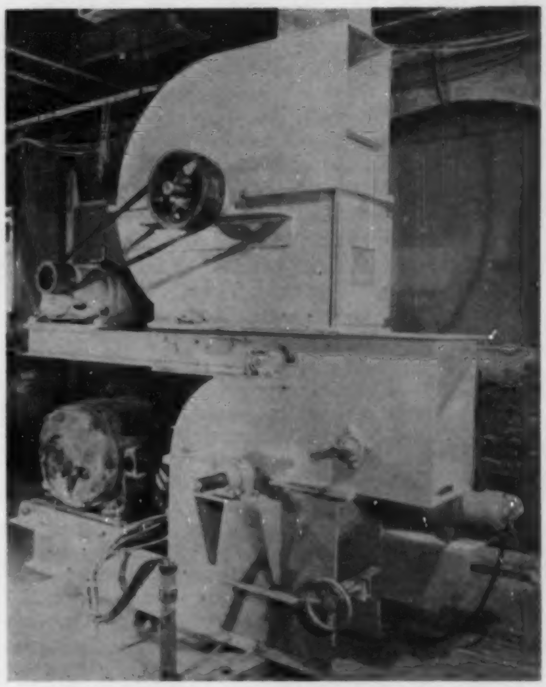


Fig. 1—Centripital classifier with 5-hp varispeed drive operating at 200 to 1000 rpm. Duplex mill below.

circulating load constantly free of superfine. The specified top size and the size that is nearly small enough continuously approach, in and out several hundred times a minute, until they eventually go in or are finally rejected and return to the mill.

The *cut size*, or size of the particles passed by the classifier, can be varied by simply changing the rotating speed of the classifier, which may be equipped with a variable speed drive for installations that process various cut sizes, in the same machine.

For large capacities, these classifiers can be designed with rotors several feet long, discharge at both ends, and drums several feet in diameter for low specific gravity and particle size range down to -1μ. The dimensions or volumetric capacity vary with specific gravity of the material and fineness of product.

Operation of this classifier is a sequence of controllable, well defined aerodynamic effects. Its classification is progressive. All superfines, that is, material finer than the specified top size particle, are instantly withdrawn. The top size is repeatedly tested before it is allowed to pass. Its precision classification depends on check and double check of the largest permissable particle.

Where only classification is called for—such as extraction of the finer fraction from material preground by some other means—the set-up remains the same except that the high-speed grinding rotor is replaced by a relatively low-speed disintegrating rotor. The principle of operation remains the same except that the oversize is progressively and continuously withdrawn from the bottom of the mill. This can be automatically controlled with little or no loss of the desired finer fraction. The grinding section of the mill was developed particularly for reduction of material in the subsieve particle range.

It will be noted, in the first place, that the rotor operates with a running clearance of 2 in. or more and with a highly fluid mill load, not over one part solids per ten parts air by volume. There is no grinding of mass material on mass material.

The rotor used is the Lykken closed-end rotor, which has a self-induced partial axle vacuum and an intense intra-blade vortex action. The rotor is operated at relatively high speed. Each time the larger particles enter the rotor and discharge from it, they are progressively reduced by impact with the rotor blade. As the material gets finer, it reduces by particle on particle attrition in the intra-blade vortex action. This is fluid energy grinding. Most of the energy applied to the rotor transforms into this intra-blade vortex action and its high intensity.

This is the most effective fine grinding principle known. There is no heat-generating mass circulation of material at ineffective energy levels.

Equally important, especially in the subsieve range, is the constant and instantaneous removal of the wanted particle size (and finer) from the grinding area. Very few seem to know and appreciate that the actual energy required to fracture the material, that is, the actual energy required to produce even the hundreds of square miles of surface in a ton of subsieve material, is only a few kilowatt-hours per ton, 5 pct or less of the power input.

That is, of course, implying that mill efficiency is ridiculously low, and it is—less than 5 pct. Most of its energy transforms into heat. It is chiefly a braking and not a breaking device. In every mill most of the energy is lost, transformed into heat by this braking effect. And most of the braking effect is due to finished product and finer in the grinding area. In subsieve grinding even a trace can reduce milling efficiency 50 pct and continue to cut it 50 pct as fines accumulate.

There are endless types of mills, some better adapted than others for a given material and product, but all depending on continuous and complete scavenging.

There is no best mill, and no best principle of mill reduction. There is only a best method of operating a mill, and that is to maintain it completely free of finished product, except that which is on its way out the fastest way it can get out, and by the shortest path.

In mill design this constitutes 90 pct of the difficulty. In the open rotor mill the fines are continuously discharged the full length of the rotor. That is why an open rotor is used, and that is why a highly fluid mill load is required.

Fine Particle Classification

Drag, eddies, and turbulence along the boundaries of classification chambers have been surmounted by new methods of air classification.

by Alan R. Lukens

FRACTIONS of powdered material may be classified according to particle size, shape, or specific gravity. Particle sizing, sometimes called *tailoring,* increases the usefulness of powdered materials in many industries. Particles larger than a specific size are detrimental in paints, paper, plastic, rubber, printing ink, and insecticides. Particles smaller than a specific size are detrimental in flour, drugs, paint, paper, and plastics. Well tailored particles are now proving valuable in the radio and television industries and in sound reproducing mechanisms. The machine tool industry finds it valuable to have exact sizing of the powders comprising cutting tools. New uses are being found for sized powdered fuels for rocket propulsion, for explosives, and for aeronautics.

Each use may present a specific problem in sizing. Dry sieving is a common method of sizing down to about 70 mesh, i.e., a little over 200μ, but sieving particles smaller than this becomes progessively inefficient if exact sizing is desired.

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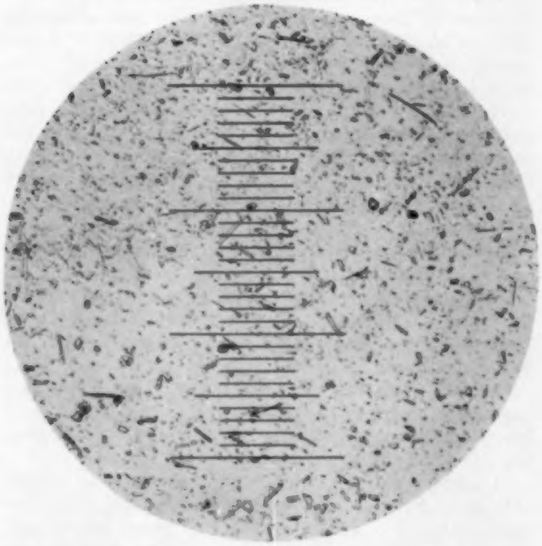
Until a few years ago, air classification seldom produced a satisfactory fractionation below 45μ (325 mesh). Only within the last five years has efficient air classification down to 10μ been available. Recently air sizing to 3 or 4μ is possible.

The present article is concerned with this new scope of efficiency and, particularly, with the advantages and disadvantages of air as the classifying fluid.

Centrifugal Classification in Water: About 20 years ago introduction of the continuous classifier, which used water, opened new areas of usefulness for more precise sizing of subsieve particles. Today several hundred thousand tons of finely particulated and sized powders are produced annually, advancing the quality of plastics, fabrics, paper, paints, rubber, insecticides, and pharmaceuticals. Productivity of wet grinding mills has been doubled by the wet centrifuge. But water classification has limits.

To obtain precise sizing the discharge from the mill is best achieved with dilution. The overflow, or effluent, becomes even more dilute in proportion to the product sought. The underflow, or rejects re-



Figs. 1 and 2—The two fractions produced from fibrous talc through the Mikroplex air classifier.

turned, is often so dry when discharged from the wet centrifuge that it must be diluted to facilitate further handling, for example, transport back to the mill.

Although this dilution of rejects involves no great expense, the dilution of the effluent does indeed involve an increasing cost, because of the present advancing cost of fuels necessary to evaporate the unwanted water about the powdered product. Time and time again, in their search for a less expensive process, manufacturers of fine particles considered using air.

Centrifugal Classification in Air: Here in the U. S. some advance in the use of gases—not only for grinding but also as a medium for classification —came with the introduction of fluid energy, or jet milling, in the 1930's.

Many people using the fluid energy mills seem unable to comprehend that the great turbulence, advantageous for reduction of particle size (grinding), is also a disadvantage in precise sizing.

A high-speed pneumatic classifier of limited throughput was developed in the 1940's, but its rotational speed and capacity were restricted because of the limitations of structural stress. This device, which would size certain products to about 25μ, suffered in precision of top sizing because of induced turbulence in the classifying chamber.

This summarizes the status of this art in the U. S. up to 1952, when a new and highly intelligent approach to efficient sizing was disclosed by Hebbs, whose invention U. S. 2,616,563 was assigned to Sharples Corp.

Four Problems of Evaluation

U. S. engineers are presently confronted by four dominant problems when considering classifiers for fine sizing:

Should present equipment be replaced?

- Do the present classifiers cushion the grinding mills?
- Is the product sufficiently free of gritty oversize?
- Are fuel expenses for evaporating too high?
- Are fuel transportation expenses too high?
- Must licensed engineers continue to supervise unnecessary heat generating plants?
- Do present pneumatic classifiers necessitate handling excessive volumes of air when fines are collected?
- Does excessive air disperse some of the fine product?

What is the best method of evaluating a classifier?

- Send a sample of the particulate that is to be sized to the manufacturer, requesting an opinion as to its sizability in his classifier. If he returns a favorable opinion, request conditions under which a pilot plant test may be made to produce a series of fractions having specific size limitations, or size limitations suggested by the classifier manufacturer.
- Request that he produce a quantity of each fraction for subsequent investigation.
- Request particle size analyses of feed, rejects, and product.
- Request an analyses of the calculated efficiency, showing the percent of feed in the product and the rejects.
- Ask how much power will be used to classify each ton of throughput.
- Request an opinion as to the potential circulating load should this classifier be integrated with the grinding or milling system.
- It is advisable to inquire how long the manufacturer has been making air classifiers. Find out how many machines he has in service and what companies are using them.
- Ask whether or not the classifier manufacturer can make a test in the size of production machine that is to be installed.
- Ask whether or not it is possible to adjust the classifier while it is in operation. This is important in maintaining continuity of circulating load.
- Does the classifier have moving walls? If so, does it infringe the Alpine patent?

When the classified products are returned how should they be evaluated?

- A small quantity placed under a microscope will not give a reliable picture of the top sizes. Instead, disperse about 10 g in water, or some other fluid that will not dissolve the powder. Use also

a small quantity of a suitable dispersing agent. Put the powder and liquids in a test tube, shake well, and allow it to settle until there is an accumulation of the larger particles in the bottom of the test tube. Decant the supernatant suspension, redisperse the sediment, and again allow it to settle. Repeat these operations several times, doing so until the supernatant liquid is reasonably clear, signifying that most of the fine particles have been removed. Then measure the sedimented large particles under the microscope. They represent the true top sizes.

- If the kind of testing equipment that will be used by a potential customer is not available, send samples to an experienced research laboratory that will make recommended tests. If the product is inferior in quality to that of competitors, go back to the company offering the classifier for further recommendations as to the machine. If the product proves superior, it may be wise to ask an experienced research organization what test data will help the sales effort. The research organization may make useful recommendations that have not previously been considered. For example, a customer may be selling his powdered material to a paint manufacturer as a pigment. This manufacturer, like many others in the paint industry, may have been forced to reduce oversize grit on his own mills to produce a smooth paint. This unnecessary milling is costly to the paint maker. It is additionally costly if during milling the color of his paint is degraded by metallic pickup from mill rolls. The paint maker may have to compensate for this by additional pigments. These add to his costs.
- Ask about the availability of newly developed fast test procedures. An experienced organization may have developed an invaluable rapid control with which to obtain an uniform product.

Will more efficient classification increase grinding mill productivity?

It has long been known that a mill operates inefficiently when cushioned by an accumulation of fines. If a ball or pebble mill is being used for dry milling, the process may be slowed by ball coating. Frequently this ball coating may be minimized, if not entirely prevented, by using a high circulating load and by using an efficient classifier, which will not return a high proportion of undesirable undersize to the mills. Likewise, if the undersize is costly, quick and efficient removel should aid mill productivity.

A New German Air Classifier

While investigating the Hebbs classifier, the author learned of a similar classifier that had been manufactured in Germany for several years. He was told that at least two U. S. manufacturers of pneumatic classifiers had gone to Germany seeking manufacturing privileges.

The Americans seeking the rights to this new machine were told that introduction of the new classifier in the U. S. market awaited issuance of U. S. patents then in process. The German manufacturers also believed that they themselves could manufacture economically and thus compete with the best the U. S. could offer. The Americans were reminded that U. S. policy favored rebuilding the economic strength of western Germany. Thus for the time being manufacture was preferred in Europe, where there was a ready market for all the classifiers that could be produced.

In 1954 the author had been retained to develop more precise classifications of a fibrous talc. Classification of this material is particularly difficult because it comprises particles that are granular, flaky, and fibrous in shape. The author had obtained test runs on the American-made devices, with one exception—that made by the Hurricane Pulverizer Co. In Augsburg, Germany, he was permitted to observe a quantity trial test run upon the same fibrous talc previously tested in the American classifiers. Results were better than any obtained on this same powder in conventional classifiers.

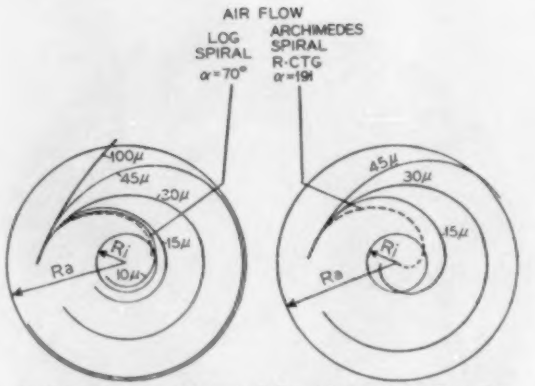


Fig. 3—At left is theoretical logarithmic spiral. Here the 100-μ particle is rejected, the 10-μ particle carried through. The 15, 30, and 45-μ particles would rotate indefinitely. At right is what Rumpf calls an Archimedian spiral, actually formed by the particle entrained air stream. Here the 45-μ particle is rejected. The 30-μ particle is critical and the 15-μ particle is carried through into fine fraction.

It was surprising to learn that techniques apparently new in America had been explored as early as 1939 by Hans Rumpf.

The research by Rumpf had been extraordinarily thorough. The reader is referred to it for details.[1] Here in part is the procedure used by Rumpf, together with his findings.

Rumpf's Research in 1939: Rumpf wisely preceded the dissertation on his research with a discussion of boundary limits of particle sizes. Instead of boundary limit, he preferred the term *trenngrenze.*

"Unfortunately," says Rumpf, "there is no agreed definition for this characteristic evolving from the separation of a powder into two sizes above and below a certain boundary limit. The most easily comprehended definition is that proposed by Dr. Theodor Eder who says, 'The boundary size (trenngrenze) is the particle size at which the probability of a particle entering either the fine or coarse fraction is similarly great.'" Rumpf continues, "Necessary computations are involved and difficult. For everyday work, an understanding of close sizing may be clarified by considering other definitions which have been given to this trenngrenze, i.e., boundary limit." These definitions are: double point, largest in fine fraction, residue from sieving of fines, point of unwarranted stray particles in fines, and percentage of coarse in fine or vice versa.

Rumpf also stated, "Whichever definition is used should always be specified. This is particularly essential when considering particulates which have a shape varying greatly from a sphere."

Rumpf devised a pneumatic classifier to investigate the conditions of particle separation that took place in a cylindrical chamber of varying diameter and upper and lower walls of varying distance apart. He constructed this apparatus to test the laws pertaining to the separation by air of a finely sized

powder into two fractions above and below a prescribed *trenngrenze,* or boundary limit, with:

d = boundary limit, or *trenngrenze.*
g = force of gravity at sea level.
n = viscosity of air during test.
s = specific gravity of the powder under test.
R = outer radius of chamber.
V = volume of air per unit of time.
H = distance between upper and lower walls.
cotg. α = cotangent of the angle of the air spiral.

Rumpf established the following working formula:

$$d = \sqrt{\frac{36\pi \cdot g.n.}{s}} \times R\,x \sqrt{\frac{H}{V}}\,\text{cotg. } \alpha$$

Thus for a use of air with a given particulate the expression: $\sqrt{\frac{36\pi \cdot g.n.}{s}}$ is a constant value, K, which simplifies the formula to:

$$d\ K.R. \sqrt{\frac{H}{V}} \cdot \text{cotg. } \alpha$$

From this ideal formula the following conclusions become apparent:

R—the larger the radius the coarser the *trenngrenze, d.*

H—height, i.e., distance apart of the upper and lower walls. Having in this formula a square root functional value, *H* would have a minor influence. The larger the value of *H* (as Hebbs claimed in 1952) the coarser the particle size at the separation boundary, *d.*

V—the volume of air, being an inverse function, likewise proves to be of less effect than, for example, cotg. α, which is in direct proportion and thus is more influential. Rumpf chose this function as being easiest to control. In his test work he confirmed the influence of these factors.

In his plan of investigation Rumpf decided to use as a variable the inclination of the spiral, i.e., cotg. α, which appeared easiest to control. He would then note the interrelation of the other constants. This formula is theoretically sound for a logarithmic spiral. Such a spiral is the theoretical path of a particle in air when vortexed in a cylindrical chamber.

Theoretically the logarithmic spiral will show equilibrium for a *range* of particle sizes even when the tangential separation factor cotg. α varies.

In one of Rumpf's test chambers the radius at the exit post was 25 mm and the radius at the perimeter was 100 mm. If the law of formation of a logarithmic spiral were to be sound, then at 25-mm radius 7μ particles of the powdered quartz would be in equilibrium and at 100-mm radius, 27μ particles would also find equilibrium. Thus the *trenngrenze* would cover the wide range of sizes between 7 and 27μ.

Again, if these particles did actually follow a logarithmic spiral, then there would ensue a progressive accumulation of particles having diameters between 7 and 27μ. Theoretically particles smaller than 7μ would leave the chamber at the center exit and those larger than 27μ would be tangentially discarded towards the outer walls.

However, Rumpf found that there was a single *trenngrenze* of about 8μ. Thus Rumpf proved that when particles are entrained in an air stream, this stream of air particles follows the Archimedian spiral path rather than the logarithmic spiral path. This he proved repeatedly.

Study of Rumpf's dissertation makes it evident that he had not, at that time, devised a means of minimizing wall turbulence, which was later remedied by rotating the upper and lower walls of the cylinder. This he and Fritz Kaiser discovered in due time, and they were granted patent protection in Germany and in the U. S.

The dominating factors controlling the spiral of air and particles as discovered and disclosed by Rumpf were:

1) Steepness of the spiral path.
2) Volume of air and particles transversing the spiral in a given time.
3) Quantity of particles entrained in a given volume of air.
4) Adjustment of exit size.
5) Radius of separating chamber.
6) Introduction of secondary air through reject port.
7) Distance between upper and lower walls.

Rumpf was apparently the first to prove the principles revealed by Hebbs, i.e., that compressibility of air is most effective at *smaller radii* and that this in turn increases the radial component of the air velocity, which in turn drastically increases the force of separation. It should also be remembered that Rumpf's findings *predicted that the closer the top and bottom walls, the finer the separation.*

The Mikroplex Classifier: This classifier is described functionally in a patent issued to Hans Rumpf and Fritz Kaiser of Augsburg, Germany. Priority in use in Germany is claimed as of Oct. 1, 1948. Application for patent in the U. S. was filed Aug. 16, 1950. The title of the patent issued Nov. 16, 1954, under U. S. Patent No. 2,694,492 is "Method of and Apparatus for Classifying Materials in Liquid and Gaseous Media." Assignment of the patent was made to Alpine A-G of Augsburg, Germany. Alpine A-G, organized in 1898, has manufactured air classifiers for 20 years and grinding mills for more than 50 years.

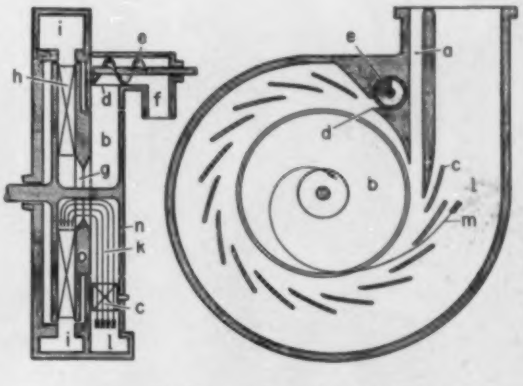


Fig. 4—Air intake at *l*, powder intake at *a*. Air spirals are formed by adjustable vanes *c*. Classification is in chamber *b*, between walls *n* and *o*. The fines exiting through *g* are induced by the fan *h*. Rejects leave at *d*, removed by screw *e*, and are discharged through *f*. Fines pass out through *i*.

In the Mikroplex classifier, disclosed by Rumpf in this patent and made by the Alpine A-G, the author found many advantages:

1) Easy accessibility to the separating chamber.

2) Easy cleaning.

3) Easy replacement of parts.

4) A clever means of keeping the narrow spaces between moving and stationary parts perfectly clean.

5) A simple mechanism whereby a series of air spiral controlling vanes could be synchronized. These vanes could be adjusted to wide latitude of degrees and locked in place while the machine was in operation.

6) Very little floor space requirement.

7) Low power demand for a given throughput of powder being classified.

8) Only 20 to 22 cu ft of air to process 1 lb of powder of about 3 sp gr.

The author was shown a means of applying linings to parts subject to abrasion. Tons of extremely abrasive particulates such as corundum were processed with negligible wear on the machine. Records confirmed use of a machine for almost one year, for this purpose, without replacement of linings.

It was demonstrated that as the oversize particles are thrown tangentially to the periphery, the direction of the stream causes them to move continuously towards the exit port. Here they cascade from vane to vane. The incoming air washes these traveling rejects repeatedly, effectively cleansing them of adhering fine particles. The walls of the classifying chamber rotate in the direction of the induced spiral of air and powder.

Returning to the U. S., the author visited a factory where 13 Mikroplex separators were being given 24-hr use 7 days a week. Only one operative was used to police all of these machines. Repair and replacement costs were negligible after 2½ years. These machines had entirely displaced the installation of a centrifugal wet classifying system devised and widely used in the U. S. because of the economy in drying, power, and labor expenditures.

Mikroplex classifiers are available in various sizes. The number designation represents the size of the classifying chamber in millimeters. Thus the 132-MP will be found to have a chamber 132 mm in diam.

These classifiers are built of steel. Castings are as smooth as many American machine surfaces. Moving parts are machined to high precision and hand finished when assembled.

A simple hand lock permits easy opening of the classifier. The operating chamber is exposed by swinging back the front mechanism, much as an old-fashioned watch is opened. The machine may be closed and locked so securely that no subsequent leakage occurs.

Although the classifying chamber of the 132-MP rotates at 12,000 rpm, if the walls of this chamber are abruptly twisted back and forth, there is no backlash.

Various feeding mechanisms are available, one with an adjustable vent. Vibration prevents arching of the feed. The feeder is motivated by the same drive that operates the classifier rotor.

The rotating walls of the classifying chamber may easily be removed. Behind these walls is the integrated fan mechanism, which furnishes the suction that in turn aspirates the air and powder through the classifying chamber.

It is possible to obtain these classifiers with stainless steel surfaces. Absolute sterility may be obtained by using steam in a closed system, which can be provided.

Explosive dusts may be classified by using an inhibiting gas in a closed installation. For additional safety the classifier can be built to withstand the pressure of an explosion.

For classifying sticky or adherent powders an ingenious *wide chamber* model is now being placed on the market.

Not mentioned in a descriptive brochure is the availability of supplementary linings, which permit classification of the most abrasive powders made by industry. In tests conducted in the author's laboratory, tons of such powders have been processed

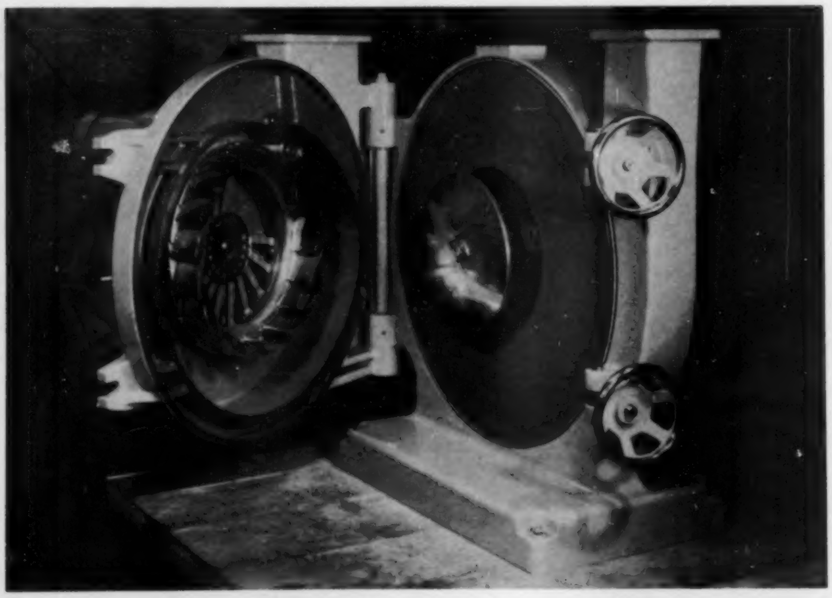


Fig. 5—Mikroplex 400 classifier opened for inspection. Classification takes place between the two polished rotating disks to the right. A particle is classified in less than 1/20 of a sec.

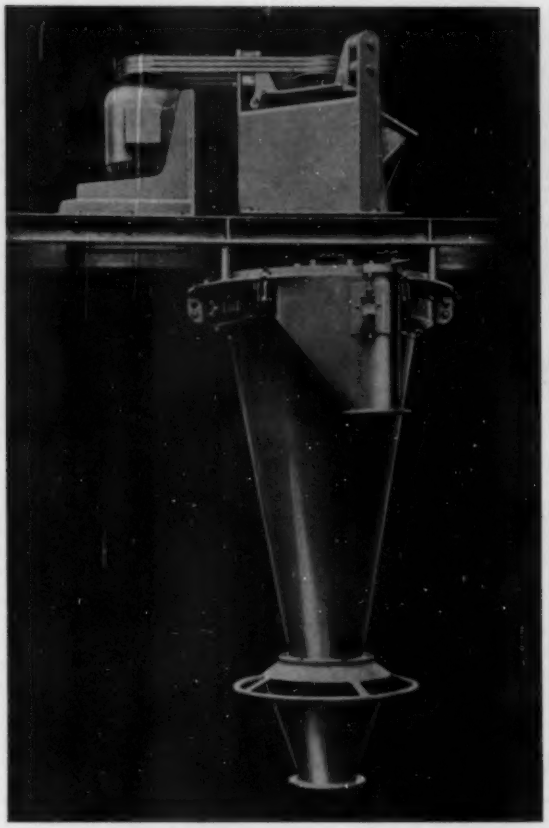


Fig. 6—Alpine Mikroplex classifier 400 MPVI. This vertical axis classifier has a separation chamber enclosed between rotating walls. The vortex plane is horizontal. Adjustment of vortex spiral is obtained by use of adjustable vanes. Feed enters from above. The top rotating wall is used as a dispersion plate.

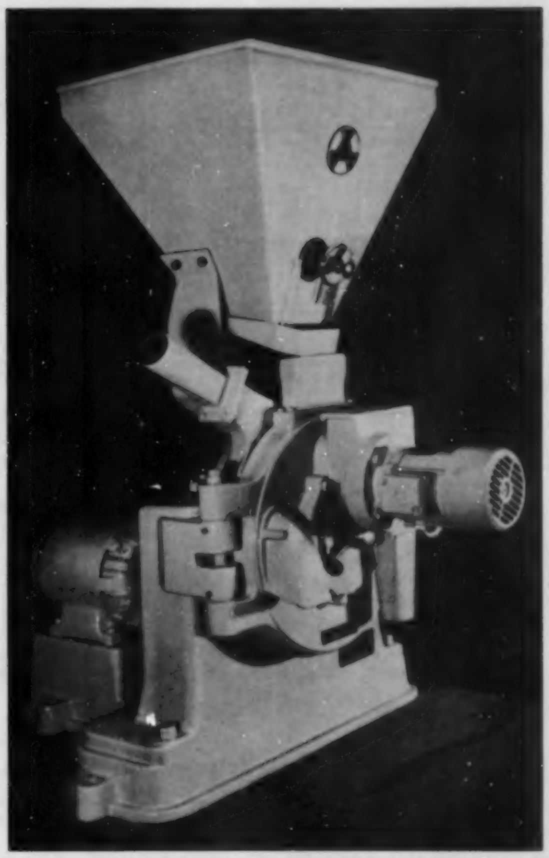


Fig. 7—Alpine Mikroplex horizontal classifier 132-MP. Moving parts are machined to high precision and hand finished when assembled. Although the classifying chamber rotates at 12,000 rpm, if the walls of this chamber were to be twisted abruptly back and forth, there would be no backlash.

without harmful wear. These same linings permit classification without discoloration of the whitest pigments.

Micrographs showing examples of separation of limestone powder, organic pigment, carbonyl iron, quartz, graphite, carborundum, cement, sugar, an adhesive pharmaceutical, and a fibrous sawdust are shown in this brochure. Classification is sharp and clean. Separations are shown to below 10μ.

The horizontal models 132-MP, 400-MP, and 800-MP are widely used in the chemical industries. Classification is made on powdered plastics, pigments, insecticides, intermediary products, pharmaceutical powders, metal powders, sugar, wheat flour, wood flour, and abrasives. These horizontal models are best qualified for the ultrafine classification. Production to 6 tph within the coarser limits has been obtained.

By increasing the size of the classifying chambers it is possible to obtain more than 6 tph. Such an increase decreases the separating power. To do so, with parallel increase in the air necessary, requires design into a recirculating air classifier. Generally, in this type of machine it has been impracticable to obtain clean separation from large air volume.

Another requirement was to obtain exact separations in the coarser field with higher throughput figures at less cost for the machinery.

The Vertical Axis Classifiers: This classifier handles up to 12 tph and discharges coarse and time fractions without an additional filter system. This model, known as the MPVI, has a separation chamber enclosed between rotating walls. The vortex plane is horizontal. Adjustment of the vortex spiral is obtained by use of adjustable vanes placed circularly. Feed input is from above. The top rotating wall is used as a dispersion plate.

Immediately below the separation chamber is the fan, connected directly to the lower wall of the chamber. This fan motivates the air stream. Underneath is the cyclone used to separate the fines from the air stream. The true solution for difficulties, typical in vertical axis classifiers, consisted in using this fan system as a two-stage air system with interconnecting chambers. The cyclone is placed between the two stages.

The cyclonic air stream passes the fan and reenters the classifying chamber as classifying air through the guiding vanes. At the same time, this air stream agitates the coarse material, which drops downwards in a helix. Thus the air has a very short cycle within both the classifying chamber and the cyclone. The stream thus maintains its rotational velocity, and this in turn reduces power demand. This air circuit, which does not waste either air or power, promoted efficient sizing of large quantities of material.

Table I. Types of Mikroplex Classifiers

Type	Separation Size, d, in Microns*	Through-put, Tph	Rotation Speed, Rpm	Power Demand, Kw, with Drive Motor
	With Horizontal Axis			
132 MP	3 to 15	0.05 to 0.2	12,000	2.2
400 MP	8 to 35	0.5 to 0.25	1400 to 2800	4 to 35
800 MP	10 to 40	1.2 to 3	1400 to 2000	12 to 35
	With Vertical Axis			
400 MPVI	15 to 60	0.5 to 2	1400 to 1800	2 to 15
1000 MPVI	15 to 90	3 to 12	560 to 1120	6 to 35
1600 MPVI**				

* Within these sizes.
** This size of machine, projected and under test, will not be put on the market until it is thoroughly satisfactory productionally. Development of high throughput machines cannot be satisfactorily projected by simple increase in dimensions.

The Wide Chamber Mikroplex: A recently developed model for classifying fast-settling and sticky powders uses the principle of the *wide chamber*, a development perfected over many years of experiment.

Conversion of Cut Size: Special tables for setting the cut size are supplied with the machine. The cut size obtained at a certain setting of the guiding vanes of a classifier depends largely on the physical characteristics of the feed material (shape and density of particles, etc.). Secondly it depends on the definition of particle size (equivalent spherical diameter, microscopic measurement of length and breadth of particles, etc.). It also depends on the definition of the cut size, which may be the average particle size of the product or the top particle size.*

* The definition of cut size used in Table I is the dimension (microscopic evaluation) of that particle for which there are equal possibilities to reach the coarse and fine side of the product. (*Trenngrenze* or boundary layer).

With materials that differ widely from spherical shape it is, of course, more difficult to define particle sizes.

Part 2 of Table I gives values found in practical separation for the different materials shown.

The influence of shape, size, and weight of the particles upon the cut size d can be represented according to Table II by the formula:

$$d = d_s K \sqrt{\frac{1}{\text{sp. gr.}}}\, d_s K$$

d_s — cut size of standard material.
d — cut size of material to be tested.
F — deviation of particle shape from sphere.

$F \times \sqrt{\dfrac{1}{\text{sp. gr.}}}$ which is constant for the same material, is represented by K in Table II.

Table II. Influence of Particle Size, Shape, and Weight Upon Cut Size

PARTICLE SHAPE	SPHERICAL	CUBICAL	ELONG CUBE	SPLINTERS	FLAKES	ACICULAR	FIBROUS
PLAN VIEW / SIDE VIEW							
DIMENSIONS REL. TO SPHERE	F = 1	F = 1.3	F = 1.9	F = 2.0	---------	---------	---------
GREATEST LENGTH	1	1.7	2.0	3.0	AVG FLAT DIAM	3-6	5-50
" WIDTH	1	1	1.5	2	3-6	1-2	---------
" THICKNESS	1	1	1	1	1	1	1
SPECIFIC GRAVITY OF THE MATERIAL (GM) (CU CM)	K = 1.4	K = 1.8	K = 2.4	K = 2.8	K = 4-8X AVG F.D.	K = 4-8X LGTH	K = 8-70X LGTH
0.5					CARBON SP. G. = 0.6 K = 4X D.	WOOD FLOUR SP. G. = 0.7 K = 4-40X LGTH	
0.7	K = 1	K = 1.3	K = 1.7	K = 2	K = 3-6X AVG D.	K = 3-6X LGTH	K = 6-50X LGTH
1.0	POLYVINYL-CHLORIDE	ORGANIC PIGMENTS SP.G. = 1.2 K = 1.3		RESIN SP. G. = 1.1 K = 3X LGTH			CELLULOSE SP. G. = 1.6 K = 30X LGTH
1.4	K = 0.7	K = 0.9	K = 1.2	K = 1.4	K = 2-4X AVG D.	K = 2-4X LGTH	K = 4-35X LGTH
2.0		LIMESTONE SP. G. = 2.7 K = 1	QUARTZ SP.G. = 2.65 K = 1		ALUMINUM BRONZE SP. G. = 2.7 K = 3XD. GRAPHITE SP.G. = 2.2 K = 4XD.	TALC SP. G. = 2.7-2.9 K = 2X LGTH	
2.8	K = 0.5	K = 0.7	K = 0.9	K = 1.0	K = 1.5-3X AVG D.	K = 1.5-3X LGTH	K = 3-25X LGTH
4.0	MOLYBDENUM DISULFIDE SP.G. = 3.12 K = 0.8	ZIRCONIUM OXIDE SP. G. = 5.57 K = 0.8	BARYTES SP. G. = 4.5 K = 0.8		MICA SP. G. = 3 K = 3X D.	MOLYBDENUM DISULFIDE SP.G. = 3.12 K = 3X LGTH - 0.7 WIDTH	
5.7	K = 0.35	K = 0.45	K = 0.6	K = 0.7	K = 1-2X AVG D.	K = 1-2X LGTH	K = 1.5-12X LGTH
8	ZINC DUST SP. G. = 6.4 K = 0.4		POWDERED IRON SP.G. = 7.8 K = 6 SPONGY IRON SP.G. = XX K = 1-2				
11	K = 0.25	K = 0.35	K = 0.4	K = 0.5	K = 0.7-1.5X AVG D.	K = 0.7-1.5X LGTH	K = 1.5-12X LGTH
18				TUNGSTEN... SP. G. = 19.2 K = 0.7X D.			

Density (specific gravity) of the classification is the actual density of the solid mineral particles and not the bulk density of the powder. When nonsolid, spongy, or cellular particles are to be classified apply the apparent gravity of a representative particle, but do not apply bulk gravity of the powder as a whole.

Reference

[1] Hans Rumpf: Zur einheitlichen Kennzeichmung der Trennscharfe Montanzeitung, Wein, September 1951, pp. 163-8.

Discussion of Air Classification Symposium

Special Note—During a general discussion period following the symposium, the three authors in general agreed that: 1) discussions of efficiency and of particle size can frequently be misleading, and therefore the method of test and language must be carefully defined, and 2) only careful pilot and commercial equipment tests, tailored to the products and conditions desired, can yield the absolute answers. Such equipment is available at the authors' laboratories.

Discussion of article by R. E. Payne

Nelson Severinghause—Cons. Quarries Corp.

Question—What are the power requirements on the Sharples classifier?

Answer—On a 10-tph (feed) classifier on talc the power requirements are approximately 40 hp, roughly proportional to feed rate.

Robert Schoemaker—Electro-Metallurgical Div., Union Carbide Corp.

Question—Can the Sharples classifier be used on metal powders?

Answer—Yes, in general the characteristic quantity in a machine of this sort is the particle diameter times the square root of the specific gravity, i.e., the higher the specific gravity the finer the possible cut-point, and you can actually obtain finer products from a denser feed. We have looked at the abrasion problem, and we have abrasion resistant liners for all the critical parts of this machine, of tungsten carbide, steel, stainless steel, stellite, etc., which are easily replaceable. We have been running a 10-tph classifier for six months now on semi-abrasive material such as talc, and on stainless steel liners have taken off less than 0.001 in. of metal. Since the material stays in the classifier for only one or less revolutions, we feel that the abrasion problem is a minor one.

Parker Pitts—Concord Mica Corp.

Question—Have you tested mica?

Answer—No, we have tried needle-shaped material, and plate-like clays. The shape certainly affects the cut point and the machine must be set accordingly. Different initial machine settings must be determined for different shaped materials for the same cut point.

James Harris—H. T. Campbell & Sons

Question—What would your efficiency be while producing a 99.8 pct — 325 mesh product? And what would the top particle size in the product be?

Answer—Very high. The efficiency does not start to drop off until we get finer than 99.99 or 99.995 pct — 325 mesh. The top size particles are frequently low-density impurities and the top particle size of the material itself would not be over 50 to 60 μ. It is hard to analyze such small quantities of material.

James Harris—H. T. Campbell & Sons

Question—What would the efficiency be on a 15μ top size product?

Answer—Subsieve particle sized terminations will vary from one analyzer to another. Roughly, the efficiency would be in the 80 to 85 pct range.

Chester Rowland—Allis-Chalmers Mfg. Co.

Question—What would the effect of the classifier be when in closed circuit with, say, a ball mill?

Answer—The Sharples classifier would cause a marked reduction in the circulating load, from 10/1 to 20/1 for a fine product or 2/1 for a coarse product, we would now expect only 20 to 50 pct circulation. This in general raises the grinder efficiency and is desirable.

Mr. Hobby—Pittsburgh Plate Glass Co.

Question—Have you operated on a hammer or roller mill feed with, say, 40 pct —325 mesh?

Answer—Yes, as low as 10 pct —325 mesh. The characteristic of the classifier is that the cut-point is completely independent of feed rate or feed particle size distribution.

Mr. Andrews—General Aniline Dye Corp.

Question—How do you adjust the cut-point?

Answer—There are two factors, the angle of the outside vanes (an easy setting) and the rotational speed of the walls, which requires a drive speed change. These must both be changed in the proper ratio. Air throughput stays constant for a given capacity.

(Continued)

A. L. Hall—Cabot Minerals Div., Cabot Carbon Co.

Statement—On pilot plant work we find that open circuit grinding and/or open circuit classification give entirely different product particle size distributions than when both are in closed circuit. It is our feeling that the air classifier people could greatly assist their potential customers if they had closed circuit equipment in their laboratories.

Retort (by Payne)—Would you not need many different types of grinding mills? As the classifier efficiency increases, the circulating load decreases, and the problem becomes one more of milling.

Comment (by Hall)—Naturally it would be nice to have several types of mills. Overall though, it is our opinion that the classifier is of prime importance and that the general effects and data from closed circuit testing would be extremely helpful.

Comment (by Payne)—You have a very real problem here. It is true that as classifier efficiency is increased, over-grinding is minimized, and a minimum of superfines (while still having a product below the desired cut-point) results. We *will* undertake a closed circuit grinding system in our pilot plant.

Discussion of article by W. H. Lykken

Wallace Jarman—Separations Engineering Corp.

Question—How rapidly do the disks of the Hurricane wear?

Answer—The wear on the Hurricane disks is practically none.

Joseph Harris—Gamble Corp.

Question—Can you cite some specific actual operating data for the Hurricane operations?

Answer—Being a consultant for Pillsbury Mills, I am not free to mention cut points or percentages, however, 40 tph is typical on flour.

David Munsell—G. L. Cabot Inc.

Question—Could you give any cut-points for other materials such as talc?

Answer—On talc we have cut at 1μ, with a 50 to 60 pct extraction, while feeding at 2 tph.

Harry Kidwell—Reduction Engineering Corp.

Question—By what method does that talc test 1μ, and what was the top particle diameter?

Answer—One micron by Fisher subsieve sizer and a substantial top size of 20μ.

Parker Pitts—Concord Mica Corp.

Question—Are you able to get a clean separation on a flaky material such as mica?

Answer—Our experience with materials other than flour and talc has been quite limited, and since 1951 patent problems, particularly in foreign countries, have limited our use of advertising and publishing of papers, etc.

Fred Kuehl—International Talc Co.

Question—What is your maximum allowable feed size?

Answer—This is not critical. We have thrown handfuls of ¼-in. gravel into the machine while feeding talc and the gravel falls out with no effect.

Mr. Stewart—U. S. Army

Question—What horsepower and rotor speed are used?

Answer—We operate at motor speed of 1750 rpm, utilizing a 100-hp motor.

A. L. Hall—Cabot Minerals Div., Cabot Carbon Co.

Question—Do you have pilot plant facilities for closed-circuit grinding and classifying tests for prospective customers of your equipment?

Answer—At the present time, no. We do have one horizontal classifier and some crushing equipment and expect soon to have a vertical machine in our laboratory.

Discussion of article by A. R. Lukens

Nathan Millman—J. M. Huber

Question—Have tests been made with Mikroplex on kaolin?

Answer—Yes. On one form we have had good results; on another which tends to pack on the side of the machine we have referred to the German manufacturers, who have another machine which they believe will work on this type.

Frank Beggs—Carbola Chemical Co.

Question—Can you give some data on the separating efficiency on talc?

Answer—Rather than go into detail now, I have a paper which discusses this thoroughly which I will be happy to send to you.

A. L. Hall—Cabot Minerals Div., Cabot Carbon Co.

Question—On the several slides showing fine and coarse Mikroplex products, were both the fine and coarse products made simultaneously in one pass on the one machine with one set of adjustments?

Answer—Yes, they were made at the same time. This is a one-operation machine; you put in one feed and get two products, a fine and a coarse. You can tailor to intermediate size, if you wish to put, say, a fine product back through the machine and then set the arrangements to get a finer cut-off.

E. Martinis—American Smelting & Refining Co.

Question—Have you run any tests on fine asbestos?

Answer—Yes and no. We have run tests on an asbestos-like material which contains fibrous particles and the difficult classification, due to the larger air/solid requirements, is obtained.

Mr. Hobby—Pittsburgh Plate Glass Co.

Question—When treating a material substantially all below 10μ and trying to cut only the +10μ material out, how does the Mikroplex work?

Answer—I do not belive you will get good classification. We prefer the feed to contain 75 pct fines and 25 pct tailings and have gone to 90 pct fines and 10 pct coarse tailings, but higher than that the efficiency drops off.

Unknown

Question—Does the Mikroplex have application in conjunction with a fluid-energy mill?

Answer—The Mikroplex has been tried briefly with a certain type of jet mill, during which minimum classifying and coarser grinding was done in the mill and the coarse particles were separated with the Mikroplex, which achieves its qualifications through lack of turbulence. I believe it can be used effectively with a jet mill.

Theodore Crane—Vanadium Corp. of America

Question—What would the effect be from tramp oversize, say −10 +14 mesh particles?

Answer—(by Mr. Norton, Dr. Lukens' associate). The small machine will take up to 1 to 1⅛ mm., the large up to 2 mm. In general a wide range of feed size is allowable.

Effect of Roasting on Recovery of Uranium and Vanadium from Carnotite Ores by Carbonate Leaching

Application of the carbonate leaching process to carnotite ores has often been limited by the difficulty of obtaining simultaneous high recoveries of uranium and vanadium. The chemical aspects of this problem are discussed and the effect of prior roasting procedures on this discovery is described.

by J. Halpern, F. A. Forward, and A. H. Ross

IN treating carnotite ores by carbonate leaching it is often necessary to subject the ore to a prior roast. Among the advantages that may result from roasting are: 1) improvement in settling and filtering characteristics of the ore; 2) destruction of carbonaceous impurities that would otherwise contaminate the leach solution and consume carbonate reagent; and 3) substantial increase in extraction of vanadium, which is usually low when the unroasted ore is leached. Recovery of vanadium may be an important factor in processing carnotite ores, and the fact that vanadium is usually recovered in good yield by acid leaching has often been an economic advantage over results obtained with carbonate leaching.

It has been found that response of an ore to carbonate leaching, following roasting, depends primarily on its composition and on the conditions of the prior roast treatment, particularly with regard to the roasting temperature, the atmosphere maintained above the charge during roasting and cooling, and the presence of certain reagents. These factors, especially the last, virtually determine the extractions of uranium and vanadium attainable by carbonate leaching, as well as the reagent consumption on leaching, whereas adjustment of the leaching conditions has only a limited influence.

These variables were examined systematically to elucidate roasting and carbonate leaching of carnotite ores and to establish a treatment yielding high recoveries of both uranium and vanadium. Tests were conducted on samples of four different Colorado Plateau ores, described in Table I.

J. HALPERN and F. A. FORWARD, Members AIME, are, respectively, Associate Professor and Head, Department of Mining and Metallurgy, University of British Columbia, Vancouver, Canada. A. H. ROSS, Member AIME, is a Consulting Engineer, Toronto, Canada.

TP 4345BD. Manuscript, Jan. 20, 1956. New York Meeting, February 1956.

Table I. Analysis of Ore Samples

Constituent	Ore A. Typical Low-Lime Carnotite, Wt	Ore B. Carbonaceous Ore, Wt	Ore C. Gypsum Containing Ore, Pct	Ore D. Typical High-Lime Carnotite, Pct
U_3O_8	0.24	0.30	1.23	0.38
V_2O_5	0.97	0.82	1.16	0.95
CaO	2.0	2.3	6.5*	8.9**
S	0.3		2.5*	0.1
Fe_2O_3	9.4		3.8	
Acid insoluble	87.0		77.0	

* Present largely as $CaSO_4$.
** Present largely as $CaCO_3$.

Roasting Procedure: The roasting tests were conducted in an electrically heated furnace in which the temperature was controlled to ±15°C. The ore (usually a 500-g charge) was ground to −100 mesh, mixed with the desired reagent, and spread in a silica tray to a bed depth of about ½ in. During the roast the charge was rabbled at intervals of 15 min. To provide free access of air the furnace was usually ventilated; however, in a few experiments a controlled atmosphere such as carbon dioxide or hydrogen was maintained above the charge.

Leaching Procedure: The pulp, generally comprising 500 g of ore and a 2000-cc solution containing 80 g per liter of Na_2CO_3 and 20 g per liter of $NaHCO_3$, was agitated in an autoclave at 120°C, under 30 psi O_2. Samples were withdrawn periodically, and the solution and ore residue were analyzed. Under the conditions employed, maximum attainable extractions were generally achieved in 4

The Transactions papers appearing in MINING ENGINEERING during 1957 will subsequently appear in AIME Transactions Volume 208, and may be permanently referenced to that volume.

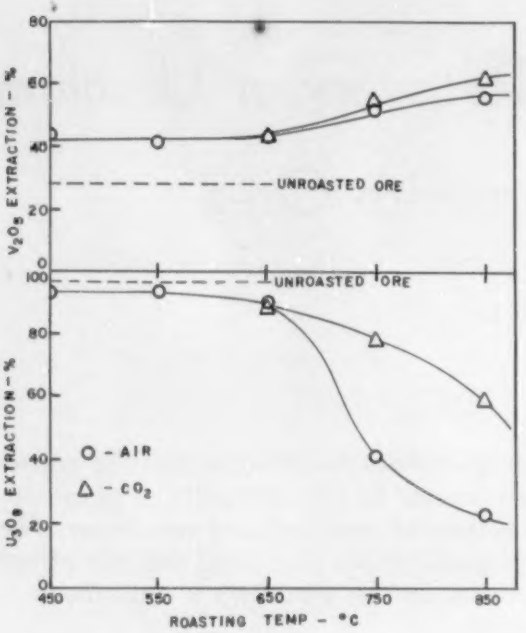


Fig. 1—Effect of roasting in air and in CO_2 on uranium and vanadium recoveries from *D* ore. Leaching 6 hr at 120°C with 8 pct Na_2CO_3, 2 pct $NaHCO_3$ solution.

to 6 hr. Final extractions were calculated, in all cases, from U_3O_8 and V_2O_5 analyses of the washed tailings.

Analytical Procedures: Ores, tailings, and solutions were analyzed for U_3O_8 with an MIT Model III Fluorimeter.[1]

V_2O_5 determinations were made by titration with standard $FeSO_4$.

Solutions were analyzed for $CO_3^=$ and HCO_3^- by potentiometric titration with standard HCl to consecutive end points at pH 8.2 and 4.0, correcting for the formation of the $UO_2(CO_3)_3^{\equiv}$ complex.[5]

Results and Discussion

Composition of Carnotite Ores: The ratio of V to U in carnotite ores is often higher than the theoretical value of 1:1 (about 0.3 parts V_2O_5 to 1 part U_3O_8 by weight) expected for the mineral carnotite ($K_2O \cdot 2\ UO_3 \cdot V_2O_5 \cdot 3\ H_2O$). Thus so-called carnotite ores generally contain other vanadium minerals, notably roscoelite ($3\ (Al, V)_2\ O_3 \cdot K_2O \cdot 18\ SiO_2 \cdot 2\ H_2O$), which often accounts for most of the vanadium present in the ore. Reference has also been made to the presence of vanadyl vanadates and calcium in some ores.

Other uranium minerals, including primary minerals such as pitchblende (U_3O_8) may also be present, although normally to a minor extent.

Among the gangue minerals that play a particularly important role during roasting of carnotite ores, silica is usually present and normally constitutes the major gangue component. Calcite may be present, sometimes in very high concentrations (as in ore *D*). Many ores also contain some reducing carbonaceous matter. Sulfides are rarely present in appreciable amounts. Gypsum sometimes occurs as a major constituent, as in ore *C*, which was tested during the course of this work.

Carbonate Leaching of Unroasted Ores: The mineral carnotite is readily soluble in solutions containing carbonate and bicarbonate salts, according to the following reaction:

$$K_2O \cdot 2\ UO_3 \cdot V_2O_5 \cdot 3\ H_2O + 2\ CO_3^= + 4\ HCO_3^- \rightarrow$$
$$2\ VO_3^- + 2\ UO_2(CO_3)_3^{\equiv} + 2\ K^+ + 5H_2O. \quad [1]$$

Thus high uranium recoveries (90 to 100 pct) are generally obtained on leaching the raw ores, under most conditions, see Table II, provided that most of the uranium is present as carnotite. If other uranium minerals such as uraninite or pitchblende are present, or if the ore contains sulfides or carbonaceous matter that can reduce uranium, it is important to conduct the leach under oxidizing conditions to effect optimum recoveries. Presence of air or oxygen under moderate pressure has been found particularly effective in insuring complete oxidation and solubilization of the uranium.[8-8] Pitchblende, for example, reacts under these conditions as follows:

$$U_3O_8 + \tfrac{1}{2}\ O_2 + 3\ CO_3^= + 6\ HCO_3^- \rightarrow$$
$$3\ UO_2(CO_3)_3^{\equiv} + 3\ H_2O. \quad [2]$$

The leaching tests reported here were all conducted at 100° to 120°C, under 30 psi O_2, to effect the maximum attainable uranium extraction for each roast treatment.

Although any vanadium which is present in the form of carnotite also is dissolved readily by carbonate-bicarbonate solutions, other vanadium minerals such as roscoelite, which often account for most of the vanadium in carnotite ores, are not readily attacked. Thus vanadium recoveries on leaching raw carnotite ores are often low, ranging from 10 to 40 pct for most ores (see Table II). Usually the amount of vanadium that dissolves is related to the uranium content of the ore, although some variation in this ratio is observed for different ores, the value often being higher than the theoretical *carnotite ratio* of 1:1. In any case, the amount of vanadium that can be extracted by carbonate leaching apparently is determined by the mineral constitution of the ore and is usually insensitive to variations in leaching conditions such as temperature, oxygen pressure, and solution composition.

To achieve substantially increased vanadium recoveries it is necessary to transform the refractory vanadium minerals such as roscoelite into more soluble compounds. This transformation can be achieved only at elevated temperatures, i.e., by roasting the ore under suitable conditions and in the presence of suitable reagents. The difficulty of finding an effective roast treatment for this purpose is enhanced by the fact that such treatments often result in the insolubilization of some of the uranium. The following sections are devoted to a discussion of the effects of roasting carnotite ores in the presence of different reagents.

Roasting Siliceous Ores: With ores such as *A* and *B*, which are largely siliceous and which do not contain appreciable amounts of reactive calcium and sodium salts, it is generally found that roasting at temperatures up to 850°C has only a moderate effect on subsequent uranium or vanadium recoveries (Table II). A slight reduction in uranium extraction usually results, probably owing to the formation of uranyl silicates, a number of which are known—for example, $UO_3 \cdot 7SiO_2$. However, these do not

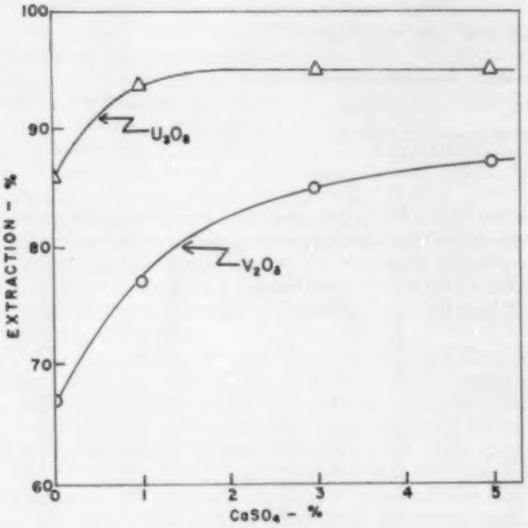


Fig. 2—Effect of $CaSO_4$ addition during roasting on the recoveries of uranium and vanadium from *A* ore. Roasting: 3 hr at 850°C. Leaching: 6 hr at 120°C with 8 pct Na_2CO_3, 2 pct $NaHCO_3$ solution.

appear to form readily by direct reaction between uranium salts and silica in the solid phase.

There is some indication that during roasting a reducing atmosphere such as H_2 or CO is detrimental to subsequent uranium recoveries. Apparently the tendency for uranium, in the reduced state, to form refractory insoluble compounds is greater than in the oxidized state, and roasting should preferably be carried out in an oxidizing atmosphere. This factor may account for the pronounced decrease in uranium extraction that results from roasting the *B* ore, see Table II, since this ore contains appreciable amounts of reducing carbonaceous matter. With this ore it was found particularly important to provide free access of air during the roast.

Roasting in the Presence of $CaCO_3$ and CaO: $CaCO_3$, which is nearly neutral in reaction, does not appear to react with uranium and vanadium minerals during roasting and does not affect the subsequent extractions to any extent. However, if the roast is conducted at temperatures above 650°C, $CaCO_3$ undergoes rapid decomposition to CaO as follows:

$$CaCO_3 \rightleftharpoons CaO + CO_2. \qquad [3]$$

The CaO that is formed apparently reacts with the uranium, forming calcium uranates, calcium uranyl silicates, etc., which are not readily decomposed by carbonate solutions. This is illustrated by the results of run B-36 (Table II) in which ore *B* was roasted at 850°C following addition of 10 pct $CaCO_3$, resulting in reduction of the U_3O_8 recovery from 73 to 17 pct. Further results illustrating this effect were obtained by roasting *D* ore, which contains about 15 pct $CaCO_3$ in its natural state (see Table II and Fig. 1). Roasting above 650°C resulted in marked decline in uranium recovery. Maintaining a CO_2 atmosphere above the charge during roasting and subsequent cooling helped to raise the uranium recoveries, as shown in Fig. 1, presumably by inhibiting decomposition of $CaCO_3$ according to Eq. 3. It is probable that higher recoveries could also be obtained by rapid quenching of the calcine in a carbonate or bicarbonate solution, since it is likely that the insoluble uranium compounds are formed, or at least stabilized, during slow cooling. Such quenching has been employed in the past[8] and found to have a beneficial effect on uranium recoveries where carnotite ores were roasted with salt prior to carbonate leaching.

The calcium vanadates, which probably form during roasting in the presence of CaO, are apparently decomposed by carbonate solutions, since the roast treatment generally results in increased vanadium extractions. This is consistent with the fact that vanadium is known to dissolve in carbonate solutions as the basic vanadate ion, VO_3^-. The following reaction would therefore be expected to take place readily:

$$Ca(VO_3)_2 + CO_3^= \rightarrow CaCO_3 + 2\ VO_3^-. \qquad [4]$$

On the other hand, it is not surprising that calcium uranates resist decomposition by carbonate or bicarbonate solution, since the soluble form of uranium in such solutions is the $UO_2(CO_3)_3^{\equiv}$ ion, i.e., a carbonate complex of the acidic uranyl ion. Conversion of the insoluble basic uranate to the soluble uranyl complex would not be expected to occur readily in basic medium.

Roasting in the Presence of $CaSO_4$: From the above considerations, it appears that to achieve high uranium and vanadium recoveries simultaneously it is desirable to roast in the presence of a calcium salt which is more acidic in reaction than CaO or $CaCO_3$ and, further, which resists decomposition to CaO at the elevated roasting temperature of about 850°C required to solubilize the vanadium. These conditions are substantially fulfilled by $CaSO_4$.

Consistent with this, it was found that addition of $CaSO_4$ to the low-lime siliceous ores (*A* and *B*) prior to roasting at 850°C resulted in high vanadium extractions on subsequent carbonate leaching, with little or no impairment of uranium extraction. Results showing this are given in Table II and Fig. 2. The amount of $CaSO_4$ required to give optimum recoveries is usually about 3 to 5 pct by weight of the ore.

With ore *C*, which contained about 15 pct $CaSO_4$ in its natural state, no further reagent addition was required. Recoveries of 97 pct U_3O_8 and 91 pct V_2O_5, compared with 94 pct and 25 pct respectively for the unroasted ore (see Table II), were readily obtained by carbonate leaching this ore following roasting at 850°C. Furthermore, mixtures of this ore with up to twice its weight of ores *A* and *B* yielded similarly high uranium and vanadium extractions following roasting without any added reagent (see Table III). Apparently a sufficient excess of $CaSO_4$ was provided by ore *C* to take care of the reagent requirements of the other ores. It is easy to see the implications of this result in regard to the possibilities for blending different ores prior to treatment.

To achieve optimum results when roasting with $CaSO_4$, it was found necessary to maintain the roasting temperature preferably between 800° and 900°C. Below 800°C, solubilization of the vanadium was generally incomplete, while roasting above 900°C usually resulted in some impairment of the uranium recovery.

As might be expected for a reaction involving solids, it was found that very fine grinding of the ore prior to roasting, preferably to −100 mesh, was

Table II. Results of Roasting and Leaching Tests

Charge: 500 g of ore (−100 mesh); 2000-cc solution
Temperature: 120°C
O_2 Partial Pressure: 30 psi
Leaching Time: 6 hr
Initial Composition of Leach Solution: Na_2CO_3—80 g per liter, $NaHCO_3$—30 g per liter

Ore	Run No.	Roasting Conditions: Temperature, °C	Time, Hr	Added Reagent	Extraction: U_3O_8, Pct	V_2O_5, Pct	Final Solution: Na_2CO_3, G per Liter	$NaHCO_3$, G per Liter
A	A-1		No prior roast		99	17	76	23
	A-2	850	3	No reagent	86	67	76	23
	A-7	850	3	1 pct $CaSO_4$	94	77	77	25
	A-6	850	3	3 pct $CaSO_4$	95	85	73	20
	A-5	850	3	5 pct $CaSO_4$	95	87	72	21
B	B-33		No prior roast		96	22	58*	41
	B-38	850	3	No reagent	73	45	78	20
	B-53	850	3	3 pct $CaSO_4$	92	70	74	21
	B-46	850	3	10 pct $CaSO_4$	83	71	64	22
	B-36	850	3	10 pct $CaCO_3$	17	78	76	17
	B-51	850	3	7 pct $CaCl_2$	0	67	78	24
C	C-1		No prior roast		94	25	30**	5**
	C-65	850	3	No reagent	97	91	65	25
D	D-1		No prior roast		97	28	68	28
	D-20	550	3	No reagent	94	42	72	27
	D-23	850	3	No reagent	24	55	80	0
	D-2	850	3	5 pct $CaSO_4$	26	60	80	0
	D-8	850	3	15 pct H_2SO_4	74	47	58	22
	D-9	850	3	15 pct FeS_2	85	64	71	24
	D-17	850	3	15 pct $FeSO_4$	55	63	89	5
	D-18	850	3	15 pct Na_2SO_4	8	38	100	0

* Large Na_2CO_3 consumption due to reaction with carbonaceous matter, which is destroyed on roasting.
** Initial solution composition: Na_2CO_3—50 g per liter · $NaHCO_3$—O.

advantageous and resulted in higher recoveries of both uranium and vanadium.

The nature of the compounds formed when uranium and vanadium minerals are roasted in the presence of $CaSO_4$ is not clearly understood. It is probable that among the insoluble uranium compounds that form on roasting in the absence of this reagent are calcium uranates and uranyl silicates, of which the simplest proto-types are $CaUO_4$ and UO_2SiO_3, respectively. For reasons of chemistry involving relative basicities of the different oxides and relative stabilities of the various compounds they can form, it is to be expected that $CaSO_4$ would lead to decomposition of the above compounds, and the formation of more soluble uranium compounds, through reactions such as the following:

$$CaUO_4 + CaSO_4 + 2\,SiO_2 \rightarrow UO_2SO_4 + 2\,CaSiO_3 \quad [5]$$

$$UO_2SiO_3 + CaSO_4 \rightarrow UO_2SO_4 + CaSiO_3 \quad [6]$$

It is probable that the compounds actually formed are of more complex constitution than those represented above, but their chemical properties might be expected to be similar. Vanadium minerals probably undergo analogous reactions in the presence of $CaSO_4$, since they are also solubilized by roasting with this reagent, possibly through formation of intermediate calcium vanadate or vanadyl sulfate compounds.

Although it should be possible to substitute other reagents for $CaSO_4$ and achieve a similar effect, the above considerations suggest that such reagents would have to meet the following requirements, all of which are fulfilled by $CaSO_4$:

1) The reagent must be somewhat acidic in reaction.

2) It must be capable of forming stable uranyl salts as well as stable silicates.

3) It must be stable up to the roasting temperature of about 800°C.

Sodium salts would not be expected to be suitable, since they do not fulfill conditions 1 and 2. They are generally neutral or alkaline in reaction and give rise to sodium silicates that are probably less stable than the corresponding calcium salts. Experiments with Na_2SO_4 and NaCl have shown these reagents to be ineffective in solubilizing uranium. Where roasting with NaCl has been used commercially in conjunction with carbonate leaching to raise vanadium recoveries,[8] it has been found necessary to control roasting conditions critically and to quench the calcines rapidly in Na_2CO_3—$NaHCO_3$ solutions in order to maintain high uranium extractions.

Table III. Roasting and Leaching of Ore Mixtures

Roasting Conditions: 3 hr at 850°C; no added reagent
Leaching Conditions: 6 hr at 120°C; 30 psi O_2; 20 pct solids
Initial Solution Composition: Na_2CO_3—80 g per liter; $NaHCO_3$—20 g per liter

Run No.	Composition of Mixture: Ore A, Wt Pct	Ore B, Wt Pct	Ore C, Wt Pct	Extraction: U_3O_8, Pct	V_2O_5, Pct	Final Solution: Na_2CO_3, G per Liter	$NaHCO_3$, G per Liter
A-2	100	—	0	86	67	76	23
A-3	50	—	50	95	90	71	22
C-65	0	—	100	97	91	65	25
B-38		100	0	73	45	78	20
B-43		67	33	91	88	72	21
B-4		50	50	91	83	70	23
B-44		33	67	93	89	65	23
C-65		0	100	97	91	65	25
M-2	33.5	33.3	33.3	93	75		

$CaCl_2$ and $FeSO_4$, which probably fulfill conditions 1 and 2 and might be expected to resemble $CaSO_4$ in their chemical effects, are not suitable since they decompose at temperatures well below 800°C. Attempts to substitute these reagents for $CaSO_4$ were not successful (see Table II).

Among the reagents that might be expected to fulfill all the above requirements and hence be effective in the roast are $CuSO_4$, $ZnSO_4$, $PbSO_4$, $CdSO_4$, $NiSO_4$, $CoSO_4$, $MgSO_4$, and $BaSO_4$. All these are acidic in reaction, decompose above 700°C, and contain metals that form stable silicates. However, the effect of these salts was not investigated, and it seems unlikely that any of them would possess advantages over $CaSO_4$.

Effect of H_2SO_4, Pyrite, Etc.: Since $CaSO_4$ obviously does not neutralize CaO, it is not surprising that its addition to ores of high lime content, i.e., ore *D*, was not effective in preventing insolubilization of uranium during roasting (run D-2, Table II). With such ores it would appear necessary to add a more acidic reagent, capable of neutralizing CaO.

This can obviously be achieved by treating the ore, prior to roasting, with H_2SO_4, which converts the CaO and $CaCO_3$ directly to $CaSO_4$:

$$CaO + H_2SO_4 \rightarrow CaSO_4 + H_2O \qquad [7]$$

and

$$CaCO_3 + H_2SO_4 \rightarrow CaSO_4 + H_2O + CO_2. \qquad [8]$$

Following such treatment the ore should respond similarly to ore *C*.

Alternatively it is possible to add a reagent that reacts with CaO under roasting conditions to neutralize the latter and convert it to $CaSO_4$. An example of such a reagent is pyrite, FeS_2, which on heating in air probably reacts with $CaCO_3$ in the following manner:

$$2\,FeS_2 + 4\,CaCO_3 + 7\tfrac{1}{2}\,O_2 \rightarrow Fe_2O_3 + 4\,CaSO_4 + 4\,CO_2 \qquad [9]$$

Thus it was found that addition of 15 pct pyrite to *D* ore prior to roasting at 850°C increased uranium extraction from 24 to 85 pct and vanadium extraction from 55 to 64 pct (run D-9, Table II).

$FeSO_4$ can also react with CaO, for example,

$$2\,CaCO_3 + 2\,FeSO_4 + \tfrac{1}{2}\,O_2 \rightarrow 2\,CaSO_4 + Fe_2O_3 + 2\,CO_2 \qquad [10]$$

but was found to be less effective than pyrite (run D-17, Table II), presumably because it decomposed more rapidly and at lower temperatures, resulting in higher SO_3 losses.

Reagent Consumption on Leaching: The amounts of carbonate and bicarbonate consumed by direct reaction with uranium and vanadium during leaching, according to Eqs. 1 and 2, are usually very small. However, total reagent consumption on leaching is normally considerably larger than this amount, because of the presence of other reactive minerals in the ore or because of reagents that have been added to the roast treatment.

Silica reacts only slowly with solutions containing carbonate and bicarbonate at temperatures below 150°C.

Mineral sulfides such as pyrite, when present in the ore, react readily with carbonate solutions, particularly in the presence of oxygen as follows:

$$2\,FeS_2 + 7\tfrac{1}{2}\,O_2 + 8\,Na_2CO_3 + 4\,H_2O \rightarrow Fe_2O_3 + 4\,Na_2SO_4 + 8\,NaHCO_3. \qquad [11]$$

However, prior roasting usually results in oxidation and decomposition of such sulfides. This applies also to organic and carbonaceous matter.

Calcium sulfate reacts with carbonate solutions as follows:

$$CaSO_4 + Na_2CO_3 \rightarrow CaCO_3 + Na_2SO_4. \qquad [12]$$

This reaction is responsible for the large reduction in the Na_2CO_3 content of the leach solution during leaching of C ore (run C-1, 65, Table II). The Na_2SO_4 generated in the solution does not appear to be in any way detrimental to the leaching.

$CaCO_3$ does not react with Na_2CO_3 or $NaHCO_3$, unless it is first decomposed to CaO by roasting. CaO reacts with $NaHCO_3$ as follows:

$$CaO + 2\,NaHCO_3 \rightarrow CaCO_3 + Na_2CO_3 + H_2O. \qquad [13]$$

This reaction is responsible for the conversion of $NaHCO_3$ to Na_2CO_3 during the leaching of *D* ore, which had been roasted at high temperatures (run D-23, 2, 18, Table II).

Precipitation of Uranium and Vanadium: After recycling, the leach solutions obtained from these ores had approximately the following analysis: Na_2CO_3, 80 g per liter; $NaHCO_3$, 20 g per liter; U_3O_8, 1 g per liter; and V_2O_5, 4 g per liter. It was found that to yield uranium and vanadium concentrates, these solutions could be most effectively treated by the following procedure:

1) The solution was stripped of both uranium and vanadium by treatment with 200 psi H_2 at 150°C in the presence of 5 g per liter of nickel powder catalyst, yielding a product comprising a mixture of uranium and vanadium oxides in combination with the nickel powder.[6, 7, 8] The Na_2CO_3 and $NaHCO_3$ contents of the solution were virtually unchanged by this treatment and the barren liquor was recycled.

2) The nickel-free product was fused with an equal weight of Na_2CO_3 and the fusion cake leached with water to extract the vanadium. The fusion cake (black cake) containing all the uranium analyzed 85 to 90 pct U_3O_8 and less than 1 pct V_2O_5.

3) The fusion filtrate was acidified with H_2SO_4 to pH 3 to precipitate the vanadium as a red cake containing 80 to 90 pct V_2O_5.

Technical Conclusions

An examination has been made of the effect of roasting carnotite ores, in the presence of various reagents, on subsequent recoveries of uranium and vanadium by carbonate leaching. An attempt has been made to account for the results in terms of the chemical reactions that occur during the roast treatment.

Roasting at temperatures of about 850°C with calcium salts, including CaO, generally increased the vanadium extraction. However, to prevent a simultaneous reduction in uranium extraction, it was found necessary to maintain the charge in an acidic condition during the roast.

With ores of low lime content, optimum results were obtained where the ore was roasted at 850°C in the presence of 3 to 5 pct $CaSO_4$. This treatment yielded extractions of 90 to 95 pct U_3O_8 and 70 to 80 pct V_2O_5 on subsequent carbonate leaching. Ores containing $CaSO_4$ may be roasted and leached without further reagent addition. Critical control of roasting conditions and quenching of the calcines are unnecessary.

With ores of high lime content, roasting at the high temperatures necessary to effect solubilization

of the vanadium resulted in a drastic reduction in the uranium recovery. For such ores, no practical treatment could be found that would yield high recoveries of both uranium and vanadium unless all the lime was first neutralized by addition of acid or pyrite. By roasting at low temperatures of 500° to 600°C, i.e., below the decomposition temperature of $CaCO_3$, uranium recoveries of 90 to 95 pct could be maintained while 40 to 60 pct of the vanadium was extracted. Such roasting is adequate to confer satisfactory settling and filtering characteristics on most carnotite ores and to destroy carbonaceous impurities.

Flowsheet Application

Fig. 3 is the type of flowsheet envisaged to give practical effect to the investigational work arising from these studies. All the types of equipment have been used for comparable purposes in various mills and refineries operating in the U. S. and Canada.

Operating Conditions and Type of Equipment: While optimum conditions will vary from ore to ore, the grinding-screening circuit prior to roasting would probably be required to produce a —35 to —65 mesh product.

In the roasting step equipment should be designed to give the best particle contact between reagent (calcium sulfate, pyrite, etc.) and the ore. The Skinner roaster has been used for many years in treating uranium-vanadium ores from the Colorado Plateau and represents a satisfactory type for this purpose.

In the case of each ore, optimum regrind mesh for calcines will depend on the usual factors of extraction efficiencies and physical properties such as filtration rates and washing efficiencies. The investigation program has suggested a range of —48 to —100 mesh.

A pulp to pulp heat exchanger of the tubular type has been found satisfactory under comparable conditions at the Beaverlodge plant of Eldorado Mining & Refining Ltd. in Canada and constitutes satisfactory equipment for this flowsheet.

In the autoclave digester section a series of vertical or horizontal stainless steel clad units is envisaged. The initial units in the series must be equipped with steam heating coils. Such equipment is used at the Fort Saskatchewan refinery of Sherritt Gordon Mines and at the Beaverlodge plant of Eldorado Mining & Refining Ltd. Pulp density in the autoclaves should be about 55 pct.

A stainless steel tubular design of heat exchanger is satisfactory for use on pregnant liquor.

Hydrogen precipitation autoclaves would be similar in general design to those used by the Sherritt Gordon Co. at Fort Saskatchewan; this equipment should be either clad or lined with stainless steel.

The nickel powder used as a catalyst in the hydrogen reduction step is strongly magnetic and can be rapidly separated from the uranium and vanadium oxides by conventional wet magnetic separation techniques.

The soda ash fusion, water leach, vanadium precipitation, fusion, and flaking equipment is standard in various mills operating in the Colorado Plateau area.

Economics: Preliminary engineering studies have indicated that the economics of this type of flowsheet in terms of capital costs, operating costs, and recoveries are equal and perhaps superior to those of alternative flowsheets now in plant scale operation where: 1) optimum recovery of vanadium is desired, and 2) a basic leach flowsheet is economically justified in preference to an acid leach from the standpoint of reagent consumption and uranium recovery.

In this connection it should be noted that although there has been a diminishing interest in byproduct vanadium recovery in mills designed in the past two or three years to treat uranium ores in the Colorado Plateau area, the flowsheet described here appears particularly well suited for recovery of uranium and vanadium from a carnotite or roscoelite type of ore with a high gypsum content.

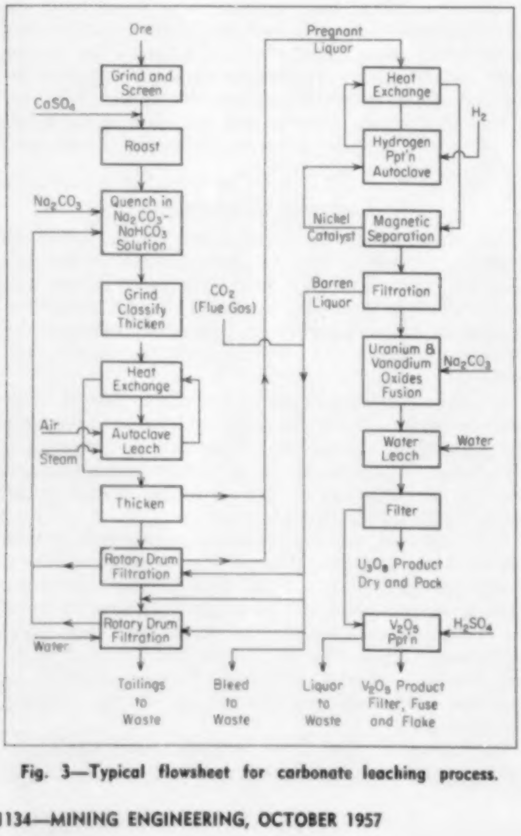


Fig. 3—Typical flowsheet for carbonate leaching process.

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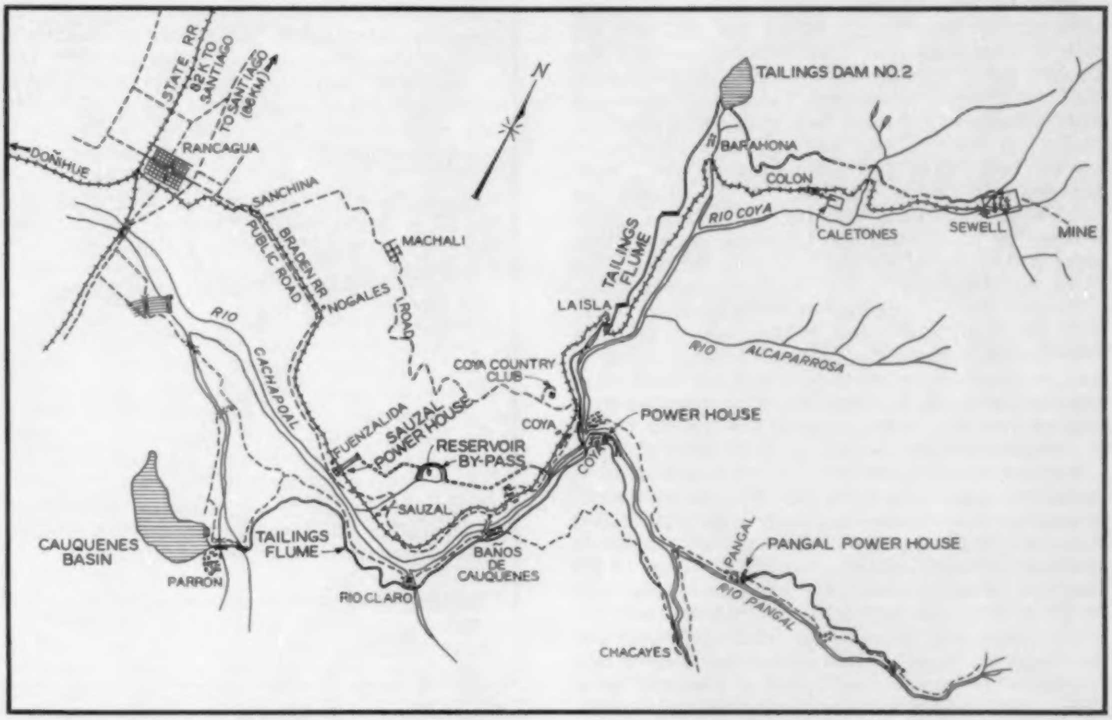

General Map of Braden Operations in Chile

Tailings Disposal At Braden Copper Co.

by R. W. Jigins

OPERATIONS of the Braden Copper Co. are in the Chilean Andes, southeast of Santiago. Most remote of the company communities is Sewell, a town of 12,000 people, 7000 ft above sea level at the junction of the Coya and Teniente rivers. Four miles down the Coya River is the smelter town of Caletones, elevation 4500 ft, with a population of about 3000. At the junction of the Coya and Cachapoal rivers 24 miles below Sewell is Coya, elevation 2500 ft, location of the company's two hydroelectric plants. The principal accounting offices; steel, iron, and brass foundries; railroad shops; and major mechanical repair and fabricating shops are at Rancagua in the central Chilean valley, 44 miles from Sewell. The company owned narrow-gage railroad connects with the Chilean national railroad at Rancagua and above Coya is the only modern means of travel.

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El Teniente mine at Sewell is the largest underground copper mine in the world. The mill, at the portál of the main tunnel, has a capacity of 34,000 tons of crude ore daily, although production is lower during the winter beacause of electric power restrictions. When operating at capacity the mill produces 2000 tpd of copper concentrates and 32,000 tpd of tailings. Tonnage of tailings is equivalent to about 100,000 tons of pulp at normal water to solid ratio.

Because the streams flowing from the Andes in the area of Braden's operations are used for irrigation, it is impossible to dump tailings directly to the river. The tailings pulp must be separated into solids and decanted clear water, and the clear water returned to the same water shed from which the original water was removed. Remaining sands and slime must be deposited in a reservoir where they will not cause trouble to farmers.

Prior to 1920 tailings were disposed of in the Coya canyon immediately below the mill site, but each of four dams that were built before 1916 was washed out in turn by the heavy floods common in the Coya River drainage area. Later three dams with total capacity of 6 million tons were constructed according to more modern design. A novel feature of the last two, called Arena and Amarga, was a floating pontoon bridge that carried the distribution launder for sand bank building. Constructed almost entirely by the spilling and shoveling method, these dams had slopes of 2.5 and 3 to 1 respectively. The dried out dams are still in good conditions today.

Beneath these three dams a single masonry aqueduct was built in the bed of the river to take the clear overflow from the decanted tailings ponds as well as the normal river flow, which increases enormously during spring run-off, often carrying millions of tons of rock and gravel. The aqueduct is 10 ft in diam and 7000 ft long on a gradient of 6 pct.

Starting in 1917 preliminary work was done to prepare a larger dam in the Barahona basin, 9 miles downslope from the mill. The original plan called for an earth dam, but this idea was abandoned in favor of sand obtained by classifying tailings in 300 balanced classifier cones. Fed through a flume line 43,000 ft long, the dam went into use in 1920.

The flume line between the mill at Sewell and the Barahona basin crossed exceptionally steep terrain and required the construction of 6500 ft of snow sheds, 19,000 ft of tunnels, and many bridges of various types. A few bridges over the deeper ravines are 150 ft high.

The crest of the Barahona tailings dam was 7200 ft long and about 210 ft high. By 1928, 29.5 million tons of tailings were built up behind the dam. In December of that year a severe earthquake broke the dam and allowed more than 9 million tons to escape through the Barahona and Coya rivers, causing loss of several lives and much property damage. After consultation with Government engineers, permission was granted to build an improved dam farther back in the Barahona basin. The new dam was started in February 1929 and was in constant use until February 1936. Today it contains 54 million tons of tailings. As capacity of this basin was limited, a new dam was begun in July 1934 at Cauquenes, 28 miles below Barahona and 37 miles from the mill. This dam was placed in service in February 1936 and is still in use.

Before the Cauquenes site was selected, many other schemes were considered and engineered. Several surveys were run, even down to the ocean, but after consideration of all factors the Cauquenes project proved the most suitable. One major consideration was the strong opposition from farmers and owners of the country the flume would cross.

The Cauquenes site was part of a prosperous hacienda, which Braden was obliged to purchase at a considerably enhanced price. The site, however, is a large, kidney-shaped natural basin roughly 3 miles long by 1½ miles wide, with a narrow outlet. Based on present dam elevation, estimated capacity of the basin is 300 million tons.

Early Dam Construction: Dam construction at Barahona is carried out by classifying the tailings in sand-settlement boxes and utilizing the coarse sands for building the dams. Remaining tailings flow by open canal to the impounding area, where they are deposited at constantly varying points to avoid build-up at any one place. Overflow towers spill the decanted water to a subterranean concrete aqueduct, which discharges by an open canal back into the Cachapoal River a few miles from the tailings dam.

Plan of Cauquenes settling basin area has an estimated capacity of 300 million tons.

The tailings flume is constructed on a 3 pct grade from Sewell to Barahona, a distance of 11 miles. From Barahona to Coya, another 11 miles, grades vary from 3 to 2 pct. In this section, to avoid excessive grades, five four-lane masonry cascade chutes were built totaling 3725 ft, with a combined fall of 1650 ft. These cascades follow a descending parabolic curve corresponding roughly to the normal discharge curve of the maximum volume of tailings carried in the flume. This helps to avoid excessive splashing and overflows.

The lower section of the line, from Coya to Cauquenes camp, is a distance of some 15 miles at an average gradient of only 0.3 pct and required construction of six large steel truss bridges, totaling 6233 ft in length, spanning canyons to avoid excessive grades loss. The largest, over the Rio Claro, is 284 ft high and 1132 ft long.

Curves in the flume, constructed with 4-ft tangential boards, are laid out on a minimum radius of 100 ft. On curves with a radius of 200 ft or more, 12-ft tangential boards are used.

In all, the disposal of tailings from the concentrator to the final area of deposition comprises the following major projects:

1) Thirty-seven miles of wooden flume line, with approximately 3x3-ft inside section, lined with concrete bottom and side liners and covered with boards at curves and through tunnels.

2) About three miles of tunnel through rock of varying types. In some sections timber sets are necessary to hold the tunnel section free from loose and falling rock.

3) One and a quarter miles of false tunnels and snowsheds to protect the flume from avalanches.

4) Two hundred and seventy-five bridges of different types, including one steel cantilever arch bridge 520 ft long; two suspension bridges with a total length of 552 ft; six large truss bridges, and 12 other smaller steel bridges of various sizes and types.

5) Eight patrol camps, where a given number of workmen are stationed, whose job it is to patrol constantly a certain section of the flume line, advising headquarters of any accident or breakout and making emergency repairs.

Operations are carried out 24 hr a day, and only three shutdowns a year are normal—January 1st, May 1st (Labor Day), and September 18th (Chilean Independence Day). On these days it is customary to make extensive repairs on the tailings flume and accessories, changing worn liners, corner strips, and drop box liners. At other times, in emergencies, the tailings must be bypassed temporarily to one or the other of the old dams. In important cases Government permission to release tailings to the river for a given period must be obtained from Santiago by telephone.

The Upstream System of Handling Tailings: The first large dam at Barahona was built on what may be called an *upstream* system, then the accepted method of depositing tailings, which were separated into sands and slimes in Callow-type classifier cones with automatic discharge. These classifiers were supported across the length of the dam by wooden trestles, which also carried the main tailings feed flume.

Starting from an earth toe the dam was built up by depositing the classifier sands to a convenient height and depositing and dewatering the slimes behind this dam. The cone supporting structure was then raised up by sections and moved back, and sand deposition was continued to complete another *lift*. As each lift was completed, the downstream face was hand-shoveled to correct grade (2.5 to 1 slope).

One disadvantage of this system was the number of men required to operate it and to maintain and continually relocate the flume, trestles, and cones. Heavy wind storms also blew over hundreds of feet of trestle, cones, and launders.

After the disastrous earthquake of 1928 it was apparent that a new system would be necessary. Temporarily, the old dam at Agua Amarga was rehabilitated and put into service while studies were made as to the ultimate solution of the tailings disposal problem.

Downstream System: To build the second dam at Barahona the *downstream* system was devised. This method started off in much the same manner, depositing coarse sand for the dam and dewatering the slimes behind. The classifier cones were raised as before, but in a *downstream* direction. In this way a dam of coarse sand was maintained, resting on similar sand, and the slime delta as built up behind followed downstream but on top of the deposited sands. This method maintains a layer of more or less impervious slimes at all times, protecting the dam from the lake, which overflows through towers at the back of the basin to a subterranean concrete aqueduct.

Another improvement developed during operation of the No. 2 dam was the change from the old type of cone classifiers to a series of tall square boxes, with sloping bottoms and screw-controlled brass sliding gates, discharging to a common launder from whence the sands were directed to any part of the dam face. Slimes as usual were deposited over the delta for settling. This method required fewer operators, although the boxes were still installed across the whole length of the dam crest and had to be raised and moved forward periodically as before.

So far the system has operated satisfactorily, but it must be noted that with this method the dam is always in operation. The exposed sand of the downstream face is subject to damage by wind and rainstorms. Sand is continually being deposited across the face and equilibrium is never established as long as the plant is operating. When it became necessary to move to the Cauquenes Basin, a better system was devised.

Sand Fill and Controlled Delta System: With the earlier methods, sands were used to form the retaining dams, and the slimes, deposited as a lake directly behind the dams, exerted a definite hydraulic pressure. This was the type of dam that failed at Barahona during the big earthquake, when the sand lifts slid downriver off their slime bases in step form. Present construction is radically different and to date has shown no signs of failure, even with very strong seismic phenomena, because: 1) only classified sands are used in actual dam building; 2) slimes are deposited and settled in a basin well removed from the dam; and 3) only a very small lake of water is maintained. This lake is some distance away and exerts relatively little pressure on the main sand dam, which is retained by a constantly rising delta of deposited tailings maintained back from the upstream face of the dams.

Present Dam Construction: For the present system a suitable site is bulldozed down to a good hardpan and opened out to give a base wide enough from upstream to downstream to allow for the designed width at optimum height, always maintaining the required slope on both faces. The slope of the upstream face is not so important, as this is eventually covered by the deposited delta. Across the longitudinal axis of the dam foundation a system of subterranean drains is built, consisting of a trench 2 ft 6 in. wide on a 0.5 pct grade discharging into a sewer type of outlet to the river. In the bottom of this trench a 6-in. concrete pipe is cemented, a ⅛-in. opening being left at the top of the joints. The whole is covered with a 6-in. layer of gravel screened between ½-in. and 2 in. On top of this, a further layer of screened rocks between 2 and 10 in. is deposited. The last layer is coarse river sand, leveled off. This built-in filter takes care of removing all toe drainage from the dam.

Parallel to the desired cross river axis two earth toe dams are built up by bulldozer to a height of 20 ft at both up and downstream limits of the base. Deslimed tailings sand is then pumped into the space below the earth dam. When this is filled with tailings sand it is bulldozed in both directions to the predetermined limit, upstream and down, until a reasonable trough is left in the center. This space is again filled by pumping in coarse tailings sand, which in turn is bulldozed to a trough in the center and refilled. As each lift is made the outside faces of the dams are corrected for slope (2.5 to 1) by bulldozing, and when finally the desired total height is reached both faces and crest are covered with a layer of soil revetment, which is then sown with coarse grass to protect it against erosion.

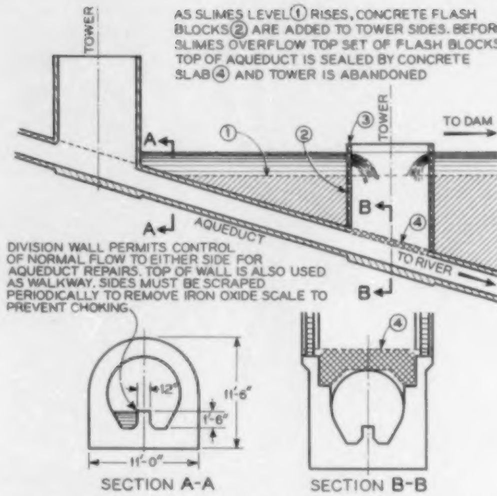
For this type of dam it is also necessary first to construct an aqueduct that will take not only the overflow of decanted water from the tailings but all the regular precipitation and flow of the watershed, up to the possible maximum of rainflooding, delivering it safely to the normal water course. This aqueduct may be built beneath the dam, rising on a continuous slope inside the basin to receive the lake overflow at suitable elevations, or it may be a tunnel driven through the surrounding hills, depending on circumstances.

Overflow towers built at various elevations discharge to the aqueduct. As the general level of delta, slimes, and lake rises the overflow control level at the tower is raised by additional concrete flashboards. When each tower reaches its height limit it is sealed off and the lake raised enough to allow it to overflow at the next tower level. During a particularly heavy rainstorm in 1953 the lake level rose 4 ft in 4 hr, and it was calculated that for a time the aqueduct discharge was 400 cu ft per sec as compared to a normal 12 to 15 cu ft per sec.

For this reason Braden Copper Co. has built an aqueduct of such strength and construction that 30 or even 50 years from now when it is buried deep in slime it will still withstand any seismic phenomena and at the same time be adequate for any possible runoff from the basin. Behind the upstream face of the dam a delta of tailings is deposited and raised gradually on a slope in advance of the general lake and slime level. As the tailings are run in, from constantly changing points, the sands settle to form the delta and the slimes run on into the lake, eventually settling to the bottom, while the clear water overflows to the outlet aqueduct. The entry point of the tailings must be constantly changed to prevent the slimes from running in directly toward the overflow tower and overflowing without settling, giving a dirty discharge to the river. This system maintains a small lake several thousand feet away and avoids pressure on the dam.

Improved Classification at the Cauquenes Site: At Cauquenes another improvement has been made in separating the sand portion of the tailings. Instead of the many small classifiers being moved around as required, there are now two long parallel desanding boxes some distance ahead of the dams and adjacent to the main tailings flume. These two boxes are built as a single concrete structure 150 ft long by 22 ft wide by 12 ft high. The bottoms slope toward a common sand discharge launder running down the center between and below them. Each box is provided with 12 bronze sliding gates for discharging the settled sands from the bottom of the box to this launder. Entrance and exit launders, controlled by gates, are provided from the main tailings launder to each of the boxes and then back to the main launder from the far end.

Operation of Cauquenes Dams: Tailings are run into one box continuously until it is filled with settled sands. Meanwhile the supernatant slimes and excess tailings continue to the main distribution launder or ditch and are deposited around the periphery of the basin for settlement. The tailings are then turned into the second box. While this is filling, the discharge gates on the first box are opened and the settled sands washed out with hoses into the discharge launder and thence to a sump serving a centrifugal sand pump, which sends them to the dam site or wherever they are required. As one box becomes full the other has just finished emptying and

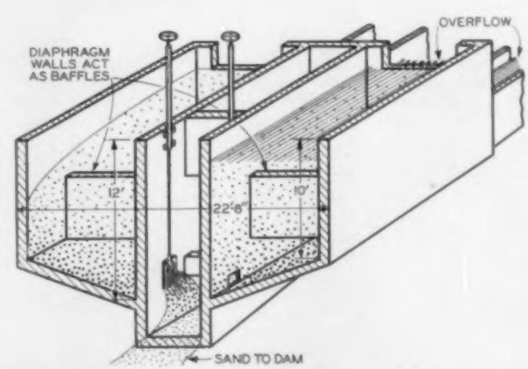


Diagrams show sequence and method of operation of Cauquenes aqueducts.

the feed is switched from one to the other steadily throughout the day. The amount of water used to wash out the boxes is maintained at a minimum sufficient to allow the sands to be pumped without difficulty through woodstaves pipelines on a practically level grade.

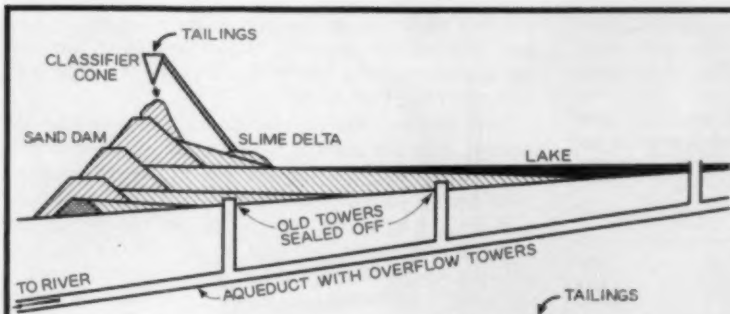
There are only two men per shift to operate the boxes; one pumpman or more, depending on the number of pumping stages; and possibly two men at the dam to divert the sand discharge as required. On the day shift a bulldozer crew will be necessary and a few men for miscellaneous jobs, but compared with the 30 to 40 men per shift required to operate the old cone classifier systems there is a great saving in labor cost. Maintenance costs are also very low, and operation is almost uninterrupted.

Maintenance of Tailings Disposal System: General repairs along the whole line are carried out continuously. Sills, collars, bracing, and tie rods on the flume are replaced constantly as necessary without interrupting operations. On the three 24-hr shutdowns during the year other repairs, such as maintenance of flume liners, are carried out in accordance with a long-time schedule. It is interesting to note that average life of the modern concrete liners is four to six years as compared to a maximum two to three years for the old wooden liners. Installed cost is practically identical.

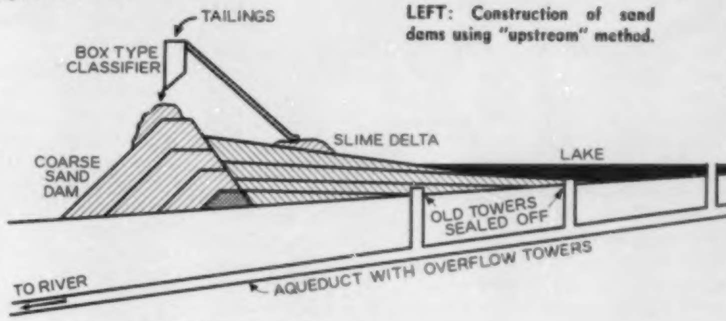


Schematic isometric section shows method of operation of desanding boxes used in tailings disposal system.

Tailings Disposal Dams and Canals

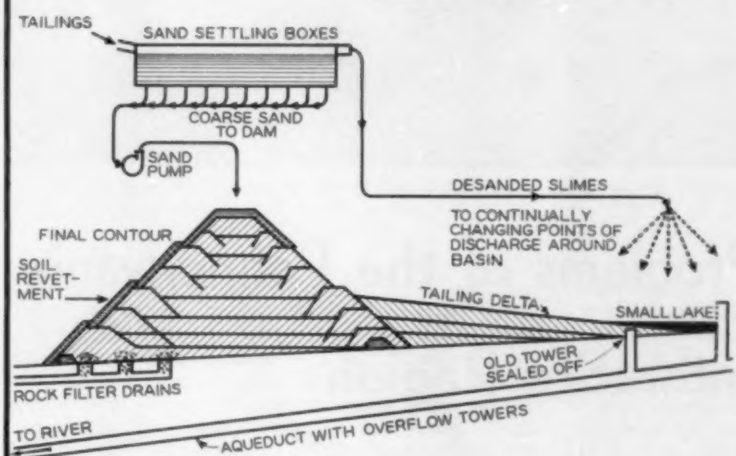


LEFT: Construction of sand dams using "upstream" method.

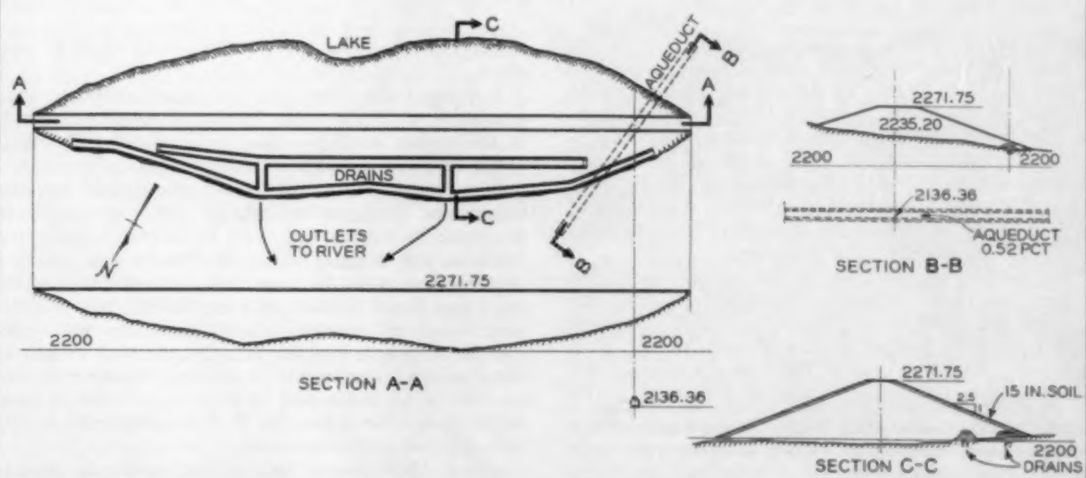


RIGHT: Construction of dams using "downstream" method.

BELOW: Sand fill and controlled delta system.



BELOW: Drainage canal design for Cauquenes dams.



Masonry cascades also require a constant maintenance program. Usually two bays per year are overhauled and repaired in each cascade. Tail boxes must be rebuilt every five or six years.

On snowsheds, some timber, roof decking, and drumsheet covering must be replaced every summer, depending on severity of the winter and the number of snowslides.

Some sections of the tunnels are in good hard rock and need no upkeep; others are in very loose ground and require constant timbering. Usually one complete set of timber is changed each day throughout the year.

Where the flume line skirts the hillsides or passes through cuts gangs are at work continually, cleaning out coronation drainage ditches, shoveling off mud slides, and jacking flume back and up on bench to maintain on grade.

In the fall there is danger from brush fires that start spontaneously on the hillsides, often covering large areas. In these cases special patrols are sent out to protect the flume and bridges. On occasion a series of auger-bit holes have been drilled in the side of the flume at strategic points, allowing part of the tailings to run downhill and help extinguish the fires.

The main drainpipes carrying the toe drainage from the dams are reamed out once a month to prevent their corroding with iron oxide deposits. For the same reason the aqueducts must be scraped and cleaned periodically. Aqueduct floors are revised and repaired every three years or so to take care of damage by rocks.

One of the chief problems in maintaining the tailings disposal system is that so much of it is difficult of access. All lumber for the flume, collars, sills, liners, and sideboards must first be cut to size and then transported to the nearest point on the railroad. From there it must be carried by mule to distribution points as close as possible to the point of ultimate use. In the tunnels in particular, this may be a mile or two away. In the final stage of transportation it is carried by workmen. Workmen also carry stones for cascade repairs and cement, sand, and gravel for cascades and flume liner repairs. In the tunnels even water for mixing concrete has to be carried in shoulderborne tanks and stored in empty 50-gal drums near the job several days before one of the major shutdowns.

Wooden trestles and bridges are repaired as necessary, and the big steel bridges are checked periodically by the engineering department.

In all, about 100 men are required to operate and maintain the Braden Copper Co. tailings disposal system. These men are distributed along the line at the different camps and moved around as the repair program requires.

Mine Water Problems of the Pennsylvania Anthracite Region

by H. A. Dierks

PENNSYLVANIA's anthracite region lies in the heart of the richest and most densely populated area of the U. S. Nearly 70 million people live within a radius of 500 miles, in which 130,000 manufacturing plants employ more than 8 million workers.

The region comprises some 3000 sq miles and contains 484 sq miles of coal measures, which folding and erosion have divided into four separate and distinct fields, known as the Northern, Eastern Middle, Western Middle, and Southern.

One million people live in the anthracite region proper, and their prosperity depends to a great extent on the anthracite industry, which produces more than $200 million in new wealth for the U. S. each year. Nearly 80 pct of the tonnage shipped out of the region is handled by nine railroads, which depend on this traffic for a large part of their revenue.

It follows that the economic health and the very existence of this industry is of real concern not only to the region itself but also to the state of Pennsylvania and the country as a whole.

Anthracite mining is one of the oldest mineral industries in America. About 1760 *burning rock* or *stone coal* was discovered on the hillsides of northeastern Pennsylvania. Within a few years a systematic search for coal bed outcrops was under way, and hand mining was begun. At first production was very limited, since anthracite was difficult to burn and was not readily accepted as a fuel. Pennsylvania anthracite is one of the densest and hardest of all coals, and its content of volatile matter is low. Consequently it does not ignite easily and it burns without smoke.

About 1808 Judge Fell of Wilkes-Barre burned anthracite in an open grate fireplace and demonstrated proper handling of an anthracite fire. From

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TP 4629F. Manuscript, Feb. 18, 1957. New Orleans Meeting, February 1957.

"Burning rock" was discovered on the hillsides of northeastern Pennsylvania in 1760 and in a few years the anthracite industry had been established.

then on anthracite found steadily increasing consumer acceptance, primarily as a fuel for home heating and for steam generation. For a considerable time it was burned as a metallurgical fuel. Mining enterprises sprang up over the entire region, and by 1837 annual production reached one million tons for the first time. Production rose steadily from year to year until a record of slightly less than 100 million tons was achieved in 1917 with employment of more than 150,000 persons. Following a short period of peak production, which coincided with the extraordinary fuel requirements of World War I, demand for anthracite began a drastic and irregular decline down to the present time. The principal cause of this steadily diminishing demand can be found in the trend toward home heating with oil or natural gas, which lend themselves more readily to completely automatic operation.

Present annual production of anthracite is about 26 million tons, and industry employment is down to 35,000. About 15 billion tons of anthracite, enough to last 300 years at the present rate of production, is still mineable and constitutes an irreplaceable and extremely valuable commodity.

Mine Water Problem: The steadily decreasing demand for anthracite has created many perplexing problems for the industry and economy of the region. Most serious of these is the mine water problem.

Reduced demand for anthracite forced curtailment of production that was accomplished with closing and abandonment of many mines, particularly those with relatively high production and pumping costs. Closing of a mine, however, is not followed by a reduction in surface water seepage. Because of breaches at various altitudes in the barrier pillars between adjoining mines, the burden of pumping the water from abandoned mines is placed inevitably on adjoining active mines.

The approximate annual cost of pumping in all mines of the anthracite region is nearly $12 million. More than 160 billion gal of water must be pumped yearly from the mines over an average hydrostatic head of 500 ft. For this task, more than 700 pumps are now installed in active anthracite mines, nearly 60 pct of them in mines of the Northern field.

The effect of this enormous pumping load on the economics of mining in the face of diminishing production is demonstrated clearly by ratios of tons of water pumped to tons of anthracite hoisted from underground operations: in 1920 the ratio was 8 tons of water to 1 ton of coal; by 1940 the ratio was 14 to 1; by 1951 it was 27 to 1; and it is now approximately 56 to 1.

The volume of water entering mine workings during and after a period of rainfall varies greatly with the different fields and even with adjoining mines in the same field. The reason is to be found in the condition of the strata, owing to the progress of coal extraction, particularly that from the uppermost beds, and to the ratios between precipitation, run-off, and infiltration as they vary for each period of rainfall in any given area. In this respect, the most important variables are severity and duration of storms; rate of evaporation, depending on temperature, humidity of the atmosphere, and dryness of the soil; transpiration, depending on the season and the amount of vegetation; status of the water

table; frost that seals crevices in the ground; and the presence of anchor ice, which seals the bottom of streams.

From year to year and from one locality to another there is a wide difference in rainfall in the Pennsylvania anthracite region. In the last 25 years, annual rainfall has varied in the Northern field from a low of 26.5 in. in 1930 to a high of 54 in. in 1945 and in the Southern field from a low of 30 in. in 1930 to a high of 60 in 1933.

Maximum volumes of rainfall are important because mine pumping facilities and sump capacities must be designed to handle that maximum inflow of water from the flash floods characteristic of the region; otherwise flooding of mine workings is inevitable.

Geologic Structures, Rainfall, and Infiltration: The geologic structure of the coal measures in the anthracite region is typically *basin and range*, and the coal beds form canoe-shaped synclines underlying the valleys between the mountain ranges. Consequently, all surface drainage from the hillsides into the valleys must flow over the coal measures. The four anthracite fields are traversed by 750 miles of rivers and streams that are part of two large drainage basins, the Susquehanna River and the Delaware River. These rivers, streams, and creeks normally lose water through their beds, and mining operations near or under these stream beds have disturbed the natural condition of the strata, resulting in increased seepage into the mines.

Since the beginning of mining in the Pennsylvania anthracite region, the general configuration of the terrain has changed, adversely affecting the original natural drainage facilities. The continued increase in the number and size of culm banks, cinder dumps, stripping pits, and stripping spoil banks has caused blockages or changes in the normal flow of surface run-off to the streams flowing on the valley floor. Cracks and caves from surface subsidence permit water to enter underground workings, and denuded woodlands drastically affect the natural control of run-off by reducing evaporation and transpiration.

Early mining operations took only the best, thickest, and most accessible anthracite beds, and little thought was given to conserving reserves for the future. Subsequent mining was made more difficult, costly, and hazardous by this lack of foresight, and, unfortunately, mining today is still conducted with the primary object of extracting as much of the anthracite beds as possible, with little attention to the ultimate effect on mine drainage. Although present mining is conducted with regard to preventing rapid and catastrophic inundation of the mines, in conformity with existing mine laws, no coordinated scheme deals with the mine water problem as it relates to mine planning, barrier pillars, and underground connections between adjacent mines.

Water that infiltrates into underground mine workings can be attributed to surface seepage, barrier-pillar seepage, or both.

Natural stream beds are pervious; however, they must continue to serve as the ultimate channels for surface run-off and to receive the discharge from mine pumps and gravity drainage from mine workings. Generally speaking, important stream beds in the anthracite region are less pervious than the terrain surrounding them, because mining operations directly underneath the stream beds have usually been conducted so as to prevent or minimize breaks in the rock strata underlying the streams. Although many small apertures in the rock strata underneath a stream bed may become sealed with silt and sludge, nevertheless large volumes of water infiltrate the mine workings through such openings under the 750 miles of stream beds that traverse the anthracite measures. It must therefore be reasoned that water pumped or drained from a mine and conducted to a natural surface watercourse does not necessarily remain confined but may contribute to the volume of water that must be pumped from some other mine farther downstream.

Barrier Pillars: A barrier pillar performs several important functions in anthracite mining operations. Its principal function, however, is to act as a dam to prevent water that accumulates in a mine from suddenly breaking into an adjacent mine.

It was not until 1891 that the Pennsylvania General Assembly passed a law making it mandatory to establish barrier pillars along the line of adjoining properties, but by that time the mine water problem was already in existence.

The barrier pillar system of the anthracite region consists of about 450 barrier pillar units, totaling 344 miles. They separate 225 mines, of which more than half are abandoned and contain more than 120 billion gal of water in underground pools.

Many of these barrier pillars were given inadequate dimensions; others have been breached by tunnels or gangways, or their stability has been impaired by the encroachment or subsidence of adjacent mine workings.

Character of Anthracite Mine Water: Although anthracite mine water is acid and destructive to iron, steel, and cast iron, it is not nearly as acid as the majority of mine waters formed in bituminous coal mines. The average acidity, determined from a great number of water samples taken from almost all the mines of the anthracite region, shows a pH factor of 3.2. The chief source of acid in the mine water is the iron sulfide (FeS) that occurs in the coal seams and in the associated rock as pyrite or as marcasite. By oxidation, in the presence of moisture, ferrous sulfate ($FeSO_4$) and subsequently sulfuric acid (H_2SO_4) in solution are produced. The former changes to ferric sulfate and forms a rust-colored precipitate, known as *yellow boy*, which so encrusts pumps and pipelines that periodic removal is necessary.

These characteristics of anthracite mine water make it imperative that pumping equipment and pipelines in contact with the water be made of corrosion-resisting material, such as bronze or stainless steel, or be protected, where feasible, with rubber or bituminous compounds.

History of Engineering Study: In the search for a satisfactory solution, the anthracite mine water problem has been the subject of many and varied studies, plans, and efforts made not only by the coal companies and the Commonwealth of Pennsylvania but most extensively by the Federal Government. Most prominent and far-reaching was the USBM study carried out from 1944 to 1954 by engineers of the Anthracite Flood-Prevention Section. In the course of this study the USBM published 25 bulletins, information circulars, reports of investigations, and technical papers on all aspects of the mine water problem, such as pumping records of all mines, underground mine water pools, condition of barrier pillars, evaluation of surface and stream

bed seepages, corrosion properties of mine water, and mapping of the buried valley of the Susquehanna River.

At the conclusion of the engineering study, comprehensive recommendations were made for a possible solution of the anthracite mine water problem. Most spectacular and ambitious was a plan for an extensive drainage tunnel system that would provide for gravity drainage of all anthracite mine water into the estuaries of either the Susquehanna River or the Delaware River. The plan called for driving a main tunnel 137 miles long, with several lateral or branch tunnels leading to the four separate anthracite fields.

In 1954 the Commonwealth of Pennsylvania established a seven-man Anthracite Mine Drainage Commission, under the chairmanship of the Secretary of Mines, to evaluate proposals for dealing with the anthracite mine water problem and, more specifically, to review the drainage tunnel scheme.

In October 1954 the Commission submitted a report to the Governor of Pennsylvania with a recommendation that the long-range tunnel plan be shelved, because of its magnitude in scope and expenditures, in favor of a short-range action plan of limited scope and cost. The plan recommended was a revision of a postwar employment proposal, formulated in 1943, providing for pumping plants in mines, and for ditches, stream bed improvement, and backfilling of stripping pits to reduce surface water seepage into the mines.

Legislative Action: On the basis of a report by the Anthracite Mine Drainage Commission to the Governor of Pennsylvania, the General Assembly of Pennsylvania and the Congress of the U. S. enacted legislation that established a State-Federal Mine Drainage Program, for which a total appropriation of $17,000,000 was to be made available. The 84th Congress enacted Public Law 162, authorizing the Secretary of the Interior to contribute an amount not to exceed $8.5 million to the Commonwealth of Pennsylvania, to be matched by Commonwealth funds of an equal amount, for projects designed for the control and drainage of anthracite mine water. Such appropriations were made contingent, however, on prior approval of individual projects by the Secretary of the Interior and on the provision that the amounts contributed by the Secretary of the Interior, and the equally matched amounts contributed by the Commonwealth, should not be used for operating and maintaining projects constructed pursuant to the Act.

The Commonwealth of Pennsylvania, by Act 82 of the General Assembly, appropriated $8.5 million, to be matched by contributions from the Federal Government, for executing such a program and directed the Secretary of Mines and Mineral Industries to construct ditches and flumes, backfill stripping pits, and crop falls; to improve stream beds for the purpose of preventing flow of surface water into mines; and to purchase and install pumps, pipes, machinery, equipment, and materials to pump water from abandoned mines. The Act also stipulates that no amounts appropriated shall be used for operating and maintaining facilities provided, or for installing any underground facilities.

Federal implementation of the mine drainage program was begun in September 1955 with establishment of a Branch of Mine Drainage in the USBM Division of Anthracite, which is charged with conducting the engineering and inspection work

This creosoted wood flume is flowing near capacity after a heavy rainfall. Mine flooding is inevitable if maximum inflow of water from the flash floods common to the area is not considered in designing mine pump and sump capacities.

involved in carrying out the Federal Government's responsibilities in connection with all matters relating to the control of drainage of anthracite mines in accordance with Public Law 162, 84th Congress. The activities of the branch involve a general but through study of the entire anthracite mine water problem and, specifically, engineering evaluation of individual programs and projects to ascertain that they are planned, designed, and executed in compliance with the spirit and specific requirements of the special act and in accordance with certain administrative procedures established between the governmental agencies.

The Commonwealth of Pennsylvania is implementing its part of the mine drainage program through the Department of Mines and Mineral Industries. A deputy secretary of this department is in charge of the activities and directs the formulation and execution of individual projects submitted by various anthracite mining companies, complete with plans and specifications attached to their petitions, for assistance in meeting the mine flooding hazard relating to a certain mine or group of adjoining mines.

In conformity with Federal Act 162 and corresponding Commonwealth Act 82, the Pennsylvania Department of Mines and Mineral Industries has full responsibility for installing, operating, and maintaining projects constructed pursuant to these Acts and must give evidence, satisfactory to the Secretary of the Interior, that projects will be so located, operated, and maintained as to provide

maximum conservation of anthracite resources and that the Pennsylvania Department will enforce effective installation, operation, and maintenance safeguards.

Flood Control Program in Action: Actual work contemplated under the State-Federal flood control program, now in progress, comprises installation or construction of three major types of facilities designed to assist the operating companies in their individual and cooperative efforts to reduce pumping costs and prevent the flooding of active mines:

1) Electrically driven deep-well pumps, of the vertical turbine type, to be placed in shafts or boreholes of idle and abandoned mines for the purpose of controlling the level of the mine water pools and preventing overflow into adjacent active mines.

2) Stream bed improvement to eliminate or at least materially reduce stream bed seepage by lining the old channels with concrete slabs or bituminous coatings, or by relocating channels onto non-caving or impervious ground or conducting the flow over broken and subsided surface in flumes or pipes made of wood, steel, or concrete.

3) Surface improvement to reduce surface seepage in certain limited critical areas by grading or ditching for unimpeded run-off and by backfilling and grading crop falls and abandoned stripping pits that are connected to underground workings.

Of the total $17 million appropriated for the joint State-Federal mine flood control program, $7.5 million has been tentatively budgeted for deep-well pump installations, $6.5 million for stream bed and surface improvements, and $3 million for backfilling stripping pits and other miscellaneous corrective measures.

Execution of the program will probably require four years, during which the Secretary of the Interior will render to the Congress, on or before February 1st of each year, a report of progress and accomplishments.

Because Act 82 of the General Assembly of Pennsylvania specifically states that the Commonwealth shall not bear the installation costs of any underground facilities, all pumping plants to be furnished under the joint State-Federal flood-control program will be installed on the surface and will necessarily be of the deep-well type.

Deep-well pumps are no novelty in the anthracite region; about 30 such pumps have been in operation at various anthracite mines during the last 15 years. They are all of the vertical turbine type, capacity ranging from 1000 to 5000 gpm for heads up to 500 ft. Submersible pumps, so far, have found no acceptance in the anthracite region, principally because up to the present they have been built only for installations that require motors of less than 450 hp.

Under the joint State-Federal mine flood control program, capacity of most of the planned deep-well pumps will range from 3000 to 6000 gpm over hydrostatic lifts of approximately 500 ft. One proposed installation, however, will consist of two pumps, each having a 3000-gpm capacity for a head of 1000 ft, requiring a 1000-hp motor.

Based on results of the USBM research study of 1954, which disclosed that all types of stainless steel containing over 12 pct Cr had adequate corrosion resistance in the most severely corrosive mine water in the anthracite region, it was considered imperative that all parts of pumps and column lines coming in contact with mine water should be made either of corrosive-resisting metal or alloy, or be protected with an acid-resisting cover material. Tests had also disclosed that the corrosion rate of the 88-2-10 bronze alloy (88 pct Cu, 2 pct Pb, and 10 pct Sn) was one-fourth that of 75-15-10 bronze alloy, which had been used almost exclusively up to that time in the pumps of anthracite mines. Consequently, all bronze parts in the deep-well pumps to be furnished under the program will be of the 88-2-10 type, designated as SAE-63. The combined erosion and corrosion resistance of stainless steels with 18 pct Cr is about 300 times higher than that of the 88-2-10 bronze alloy and 1200 times higher than that of the 75-15-10 bronze alloy. In view of these established facts, the specifications for the pumps to be purchased under the mine flood control program call for the widest application of stainless steel in all parts of the pumps that come in contact with mine water. Pump bowls are to be made of SAE-63 bronze. Column lines will be made of carbon steel but are to be lined and coated with a 3/16-in. semihard rubber compound. The flanges of the pipes cannot be covered with rubber; therefore, they will be made of stainless steel.

All planned deep-well pump installations will be designed for automatic or semi-automatic operation, actuated either by a time clock or float arrangement, or a combination of both. Operation and maintenance of the pumps will be the contractual duty of the owner of the mine where the pumping plant is located, or by the owner of an adjacent mine whose operation is protected by the facilities provided.

Before any project proposal can be accepted and approved, the petitioner must conclude an operation and maintenance agreement with the Commonwealth of Pennsylvania for a minimum period of five years, whether for a deep-well pumping plant, surface flume, or stream bed improvement project. Sixteen individual projects have so far been approved or await approval. Contracts for these projects will be awarded by the Pennsylvania Department of Mines and Mineral Industries to the lowest responsible bidder.

Conclusions

It must be emphasized that the present Federal-State mine drainage program does not completely solve the mine water problem of the anthracite region. At best it is an emergency program designed to prevent disastrous mine floods.

Should the future demand for energy derived from solid fuel increase in line with current predictions, a stronger market for anthracite is a distinct possibility. It is also expected that current research in the utilization, preparation, and mining of anthracite will bring about new markets for this commodity. Shrinking reserves of oil and natural gas will ultimately necessitate substituting other fuels if future U. S. energy requirements are to be met to the fullest extent in war and peace. It seems prudent national policy to assist the hard-pressed anthracite industry in preserving a basic natural resource that will provide: 1) a continuing source of supply for present users of anthracite; 2) fuel insurance during national emergencies; 3) insurance against the untimely ultimate depletion of domestic oil and natural gas; and 4) continued economic health of a sizable segment of the population.

Modern Grinding Plant Design in the Cement Industry

by W. R. Bendy

GRINDING is a large and costly part of Portland cement manufacture. Prior to clinkering in the rotary kiln, raw materials are ground to a fineness of 80 to 90 pct passing 200 mesh. Then, after burning and cooling, the resulting clinker is ground to about 92 pct passing 325 mesh.

In the cement industry the most favored method of grinding has always been by impact and attrition of a ball charge in a rotating mill. Other types of mills, in which materials are ground between die rings and rollers, or between die rings and large balls, are sometimes used for single-stage grinding and often for preliminary grinding followed by grinding in ball-type mills. Their efficiency is usually high, but maintenance and repair costs are high also, and the ball-type mill continues to be the most widely used.

Together with 4 pct gypsum to retard setting time, the kiln run clinker—consisting of hard, semi-fused lumps that may have been crushed to $-\frac{1}{2}$ in.—is ground to a Blaine surface area of 3000 sq cm per gram and about 92 pct passing 325 mesh. Approximately 32 kw-hr per ton, or 6 kw-hr per 376-lb barrel, are consumed in the grinding mills alone, not counting auxiliaries.

Open Circuit Clinker Grinding: Many early clinker grinding plants employed two-stage, open circuit grinding. Comparatively short mills of large diameter, loaded with balls 4 in. and smaller, first reduced the clinker to 95 pct passing 14 mesh. Tube mills of smaller diameter and greater length, loaded with balls $1\frac{1}{4}$ in. and smaller, carried out the fine grinding operation.

An early development was to assemble two or more stages of grinding in a single mill having two, three, or four compartments separated by division heads. There were difficulties in balancing the compartments, but compartment mills were popular in the 1920's because the layout was simple and there were no elevators and conveyors.

Oversize Particles: Open circuit mills, whether separate ball and tube mills or compartment mills, always encountered the problem of tramp oversize in the product. A small percentage of clinker survived passage through the first grinding stage as particles—perhaps 1/16 in. diam—that were too large for the smaller balls in the following stages to reduce to -200 mesh. When the feed rate was lowered to reduce tramp oversize, there were serious losses in grinding efficiency.

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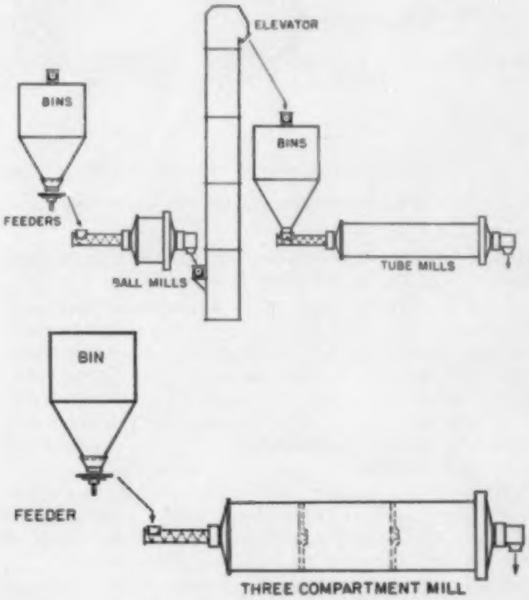
TP 4625BH. Manuscript, Nov. 5, 1956. Northeastern Mining Branch Conference, Hershey, Pa., Nov. 8-10, 1956.

Many ingenious designs were perfected to incorporate perforated screen plates or wire mesh screens in the ball mills or in the first compartments of compartment mills. Their purpose was to prevent material passing out of the first stage until it was small

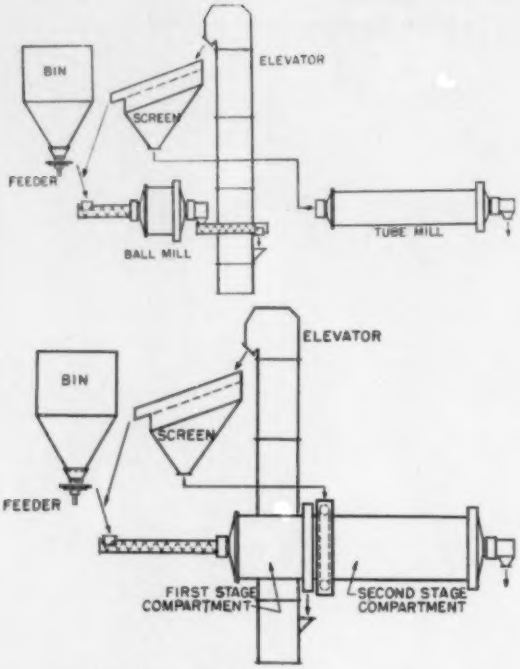


TOP: Open circuit ball and tube mills.

BOTTOM: Open circuit compartment mill.

Designer's Check List

1. Raw and clinker mills in line.
2. Traveling crane over all mills.
3. Headroom and through passageway for trucks under mills.
4. Provision for storing, loading, and weighing grinding balls.
5. Spotting provisions on mill motors.
6. Hoisting beams over elevators.
7. Unimpeded, straight-line passageways for operators.
8. Access between floors by shortest route.
9. Centralized operating controls.
10. Strategically located welding outlets.
11. Permanent vacuum cleaning system and outlets.
12. Washroom facilities.
13. Chemical control laboratory in mill room.



TOP: Ball and tube mill with vibrating screen.

BOTTOM: Compartment mill with vibrating screen.

enough. Separate vibrating screens were also used, the tailings being returned to the mill.

Closed Circuit with Air Separator: Although screens after the first stage of grinding largely solved the problem of tramp oversize, the final stage still remained in open circuit. With the passage of time, higher and higher cement finenesses were called for. Overheating and ball coating reduced grinding efficiency. It became increasingly costly to produce the new high-fineness cements in open circuit.

The solution to these difficulties was the air separator, long used in Germany before it was adopted in the U. S. The original feed enters the separator at the top and falls onto a rotating plate, where it is thrown out by centrifugal force. While moving radially outward, the particles of cement encounter a cross-flow of air impelled by the rotating fan blades. Fine particles are picked up by the air current and deposited in the outer cone, leaving by the fines spout. Coarse particles are not picked up by the air and fall into the inner cone, leaving by the tailings spout.

Although the air separator does not give a clean separation of particles finer than about 48 mesh, it does divide the mill product into fines and tailings. The fines contain more of the −200 mesh particles than the tailings, and, conversely, the tailings contain more of the +200 mesh particles than the fines. In the table in column 1 showing typical particle size distributions the tailings amount to about 1.2 times the fines. It will be noted that the tailings contain some fine material.

Particle Size Distribution in an Air Separator

Size Range, μ	Feed to Air Separator, 220 Pct	Fines from Air Separator, 100 Pct	Tailings from Air Separator, 120 Pct
+45	55	25	80
−45 +30	7	12	3
−30 +20	10	18	4
−20 +10	11	19	4
−10	17	26	9
	100	100	100

When the tailings from an air separator are returned to the mill, a circulating load is created. Although the circulating load contains a certain amount of fine particles, the material in transit through the mill is considerably coarser than in open circuit. This reduces the formation of coating on the small balls. It is also possible to draw a large quantity of air through the air separator, exhausting it to a bag-type dust collector, thereby cooling both the tailings being returned to the mill and the finished cement. Imperfect as the air separator may be as an accurate sizer of fine particles, it has the following advantages, which become more pronounced with high-fineness cements: 1) elimination of oversize particles; 2) reduction of ball coating; 3) lowering of temperature of mill and product; 4) some gain in mill output.

The aim of closed circuit grinding is to remove fine particles before power is wasted in overgrinding them. In the case of cement, it is to desired to produce surface area and a resultant plastic, workable concrete. Too great a reduction in the proportion of fine particles might be undesirable. For many years, therefore, the belief persisted that open circuit cements might have qualities superior to those of cements produced with an air separator, but repeated investigations have failed to show any measurable difference. In this respect, the inefficiency of the air separator may be a desirable attribute in grinding cement. Most U. S. cement plants now use air separators in grinding cement.

It is entirely feasible first to grind the clinker in a preliminary mill, and then to follow it with a tube mill in closed circuit with an air separator. This is done in many older installations, in which air separators have been applied to existing tube mills. Once the air separator is employed, however, no advantage can be found in doing the preliminary grinding in a separate mill. Modern installations often use the closed circuit compartment mill, which is simpler.

With an air separator it has been found equally feasible to use a short ball mill instead of a longer, multicompartment mill for grinding cement. The only requirement is that the clinker be crushed to about −½ in. so that the mill can be loaded with a graduated charge having a maximum size of ball no larger than 3 in. So far no appreciable difference has been found in efficiency or in the quality of product between the closed circuit ball mill and the closed circuit compartment mill.

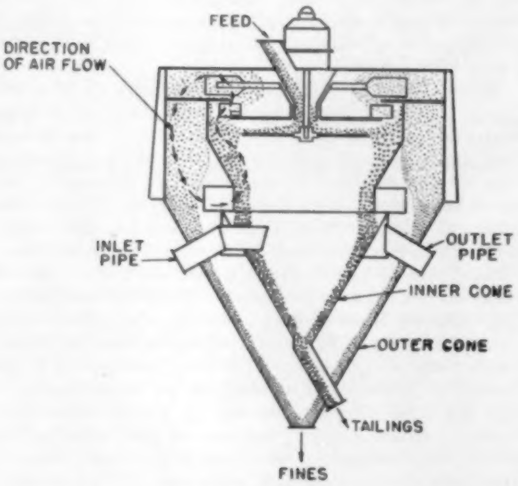
In the closed circuit compartment mill illustrated here the tons per hour of separator fines is the same as the mill feed. Separator tailings form the circulating load. Circulating load, expressed as the ratio of tailings to the original mill feed, may be calculated as follows:

$$\text{Circulating load} = \frac{\text{Separator tailings, tph}}{\text{Original mill feed, tph}} = \frac{\text{Pct fineness sep fines} - \text{pct fineness sep feed}}{\text{Pct fineness sep feed} - \text{pct fineness sep tailings}}$$

Arrangement of Units for Clinker Grinding: The lowest capital investment for a cement plant of given

Data for One 1500-Hp Clinker Grinding Mill

Output	200 bbl per hr	
Feed size	−⅛ in.	
Fineness of product	3000 SS, Blaine	
Circulating load	600 pct	
Elevator capacity	270 tph	
Air required for dust suppression and cement cooling	21,000 cfm	
Temperature of exhausted air	180°F	
Temperature of cement	200°F	
Installed Horsepower		
Mill	1500	
Excitation	50	
Separator	125	
Elevator	50	
Dust collector	60	
Cement pump	100	
Compressor for cement pump	125	
All feeding, conveying, and miscellaneous	30	
Total	2040	
Power Consumption	**Kw-Hr Per Bbl**	**Kw-Hr Per Ton**
Mill alone	6.0	32.0
Auxiliaries, less pumping	1.1	6.0
Pumping finished cement	0.8	4.1
Total	7.9	42.1
Capital investment, including all equipment, installation, building, electric wiring, transformers	$700,000	



Typical air separator.

Compartment mill in closed circuit with air separator.

Particle Size Distribution of Typical Cement

Size Range, μ*	Pct
+45	25
−45 +30	13
−30 +20	17
−20 +10	19
−10	26

* 1μ = 0.001 mm or 1/25,400 in.

size can be attained by using the fewest number of the largest units readily available. Grinding mills in the cement industry have reached the 1500-hp mark and will probably become even larger. For such a mill, the designer of a clinker grinding plant must assemble the equipment listed below. It will be understood that many other size combinations are possible:

Mill feed bins, with feeders.
A 12½x18-ft ball mill.
A 1500-hp, 200-rpm motor.
A 10-ton service crane for repairing the mill.
A 16-ft diam air separator.
A 270-tph elevator, for 600 pct circulating load.
A 21,000-cfm dust collector.

A longer mill of small diameter, with two compartments separated by a division head, may be substituted for the short ball mill.

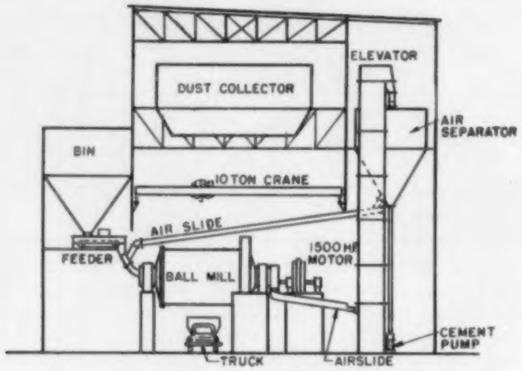
Various arrangements of this equipment are possible. If the air separator is placed near the discharge end of the mill, the elevator can be conveniently located next to the air separator. The tailings must then be conveyed back to the feed end of the mill. If the tailings conveyor is located beneath the service crane, there is some restriction to the movement of the crane hook. If it is placed above the service crane, the air separator must be installed higher.

If the air separator and the elevator are placed at the feed end of the mill, the mill product can be conveyed back to the elevator by an airslide. All auxiliary equipment is concentrated in the high section of the building at the feed end of the mill. A lower section of the building houses the mill, motor, and service crane.

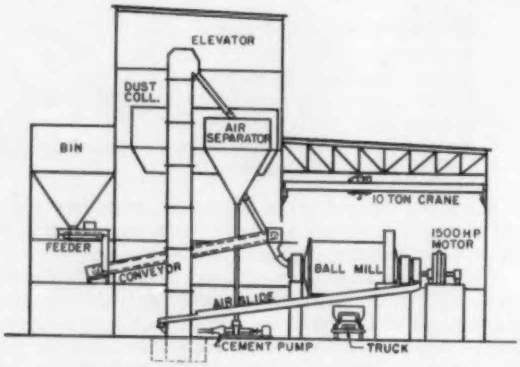
In the illustrated variation of these arrangements, the elevator is placed at the discharge end of the mill and the air separator at the feed end. The mill product must then be carried from the head of the elevator to the air separator by a conveyor located above the service crane.

Raw Material Grinding: In the cement industry raw material grinding may be carried out as part of the wet process, in which a slurry is produced for feeding to the rotary kilns, or by the dry process, in which dry pulverized raw material is fed to the kilns. The most common primary material is limestone, although cement rock, marl, or marine shells are sometimes used. Secondary materials are usually shale or clay. In the wet process they may be ground with the limestone or ground or washed separately and blended in later in the form of slurry. In the dry process they are ground with the limestone.

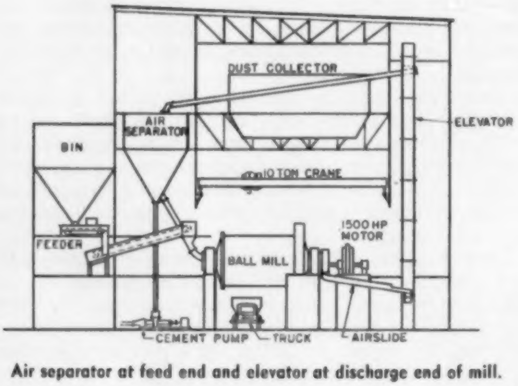
In contrast to clinker grinding, surface area is probably less important in raw grinding than elimination of all particles above a certain size (perhaps 100 mesh) that do not react readily in the rotary kiln. With open circuit grinding a fineness of at least



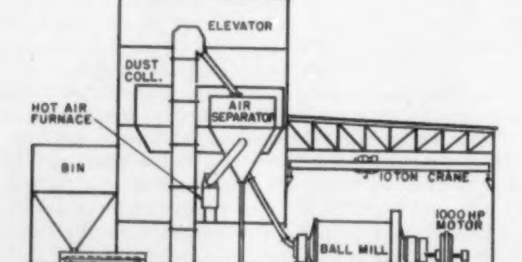
Air separator at discharge end of mill.



Air separator and elevator at feed end of mill.



Air separator at feed end and elevator at discharge end of mill.



Raw drying and grinding installation.

Data for One 1000-Hp Raw Grinding Mill

Output	200 bbl per hr (60 tph)	
Feed size	— ⅛ in.	
Fineness of product	80 pct passing 200 mesh	
Circulating load	500 pct	
Elevator capacity	360 tph	
Average moisture in raw materials	5 pct	
Air required for dust suppression and drying	18,000 cfm	
Temperature or air to separator	600° to 800°F	
Temperature of air to dust collector	180°F	
Dewpoint of air to dust collector	120°F	
Installed Horsepower		
Mill	1000	
Excitation	30	
Separator	125	
Elevator	60	
Dust collector	50	
Cement pump	100	
Compressor for cement pump	125	
All feeding, conveying, and miscellaneous	25	
Total	1515	
Power Consumption	**Kw-Hr Per Bbl**	**Kw-Hr Per Ton**
Mill alone	4.0	13.3
Auxiliaries, less pumping	1.0	3.3
Pumping raw material	0.8	2.6
Total	5.8	19.2
Capital investment, including all equipment, installation, building, electric wiring, transformers	$650,000	

90 pct passing 200 mesh is commonly used. With closed circuit grinding, resulting in fewer coarse particles and fewer ultrafine particles, finenesses of 80 pct passing 200 mesh often give a perfectly satisfactory kiln feed.

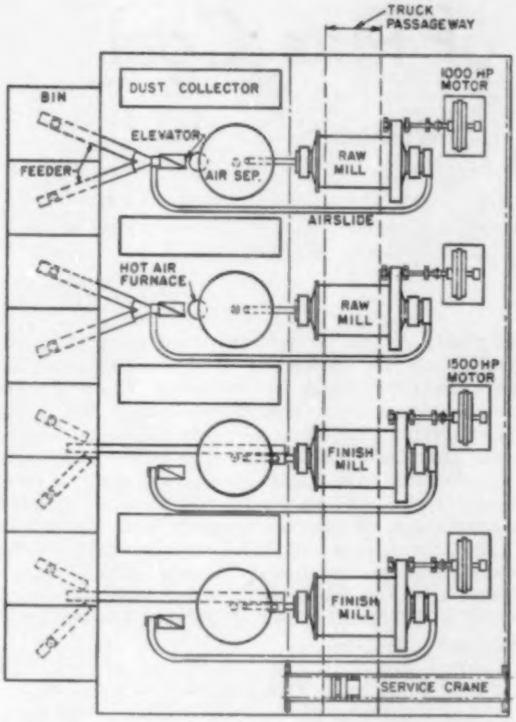
Grindability of raw materials varies widely. Compared to about 32 kw-hr per ton for clinker, raw material grinding may consume 12 to 22 kw-hr per ton, depending on the use of wet or dry grinding and on grindability of the material. Owing to loss of weight by calcination, the weight of dry raw material to be ground is about 1.6 times the weight of finished cement. Despite this, raw grinding consumes only 65 pct as much power in the process of cement making as that required for clinker grinding.

Combined Raw Drying and Grinding: The older system is to pre-dry the raw materials in rotary driers before grinding. The modern system is to flash-dry in the air separator. Both the original feed and the mill product are fed to the air separator, through which a large volume of preheated air is drawn and exhausted to a bag-type dust collector. The tailings from the air separator are practically dry and form the only feed to the mill. Enough air must be supplied to act as a vehicle for the water vapor, in order to avoid condensation in the dust collector.

Dry raw material grinding has undergone the same changes as clinker grinding in developing from two-stage open circuit to single-stage closed circuit. However, a compartment mill is not considered suitable or necessary for combined drying and grinding —the single-compartment ball mill is usually employed.

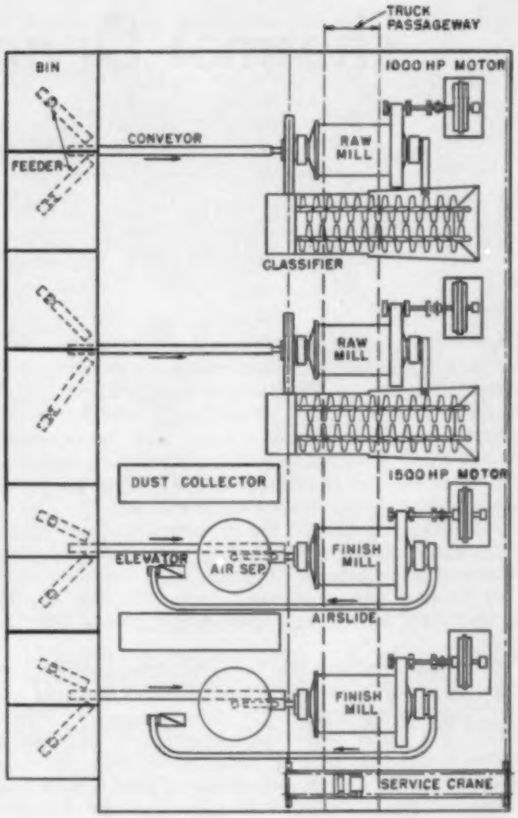
The chief difference between modern dry raw grinding and clinker grinding is that a hot air furnace must be provided to heat the air supplied to the air separator. The original feed passes through the air separator before entering the mill.

Wet Material Grinding in Open Circuit: In the cement industry wet raw grinding has gone through stages of development similar to clinker grinding. The first step was two-stage grinding in open circuit, either with ball mills followed by tube mills or with compartment mills. Then screens were added after the first stage to avoid tramp oversize particles. The



ABOVE: Combined raw and clinker grinding department dry process.

RIGHT: Combined raw and clinker grinding department wet process.



similarity ceases at this point, because in the second stage no method of classifying fine particles is very effective with the viscous slurry required for kiln feed. Thin slurry classification requires subsequent thickening, which is sometimes regarded as an undesirable complication of the process. The final stage of wet grinding has therefore remained in open circuit at many wet process cement plants, both old and new.

To obtain the lowest possible fuel consumption in the rotary kilns the slurry is ground in a highly viscous state with as little water as possible. Usual moisture content is about 35 pct, although some raw materials require 40 pct or higher. Each 1 pct of additional moisture content increases the fuel consumption per unit of output by about 1.8 pct.

Closed Circuit Wet Grinding: To close the final stage of the wet grinding circuit with a bowl, rake, or screw classifier the slurry must be diluted to about 80 pct moisture. Thickeners are needed to reach an acceptable kiln feed moisture.

In 1930 the first thin slurry, full closed circuit, wet grinding system was installed by Dewey Portland Cement Co. at Davenport, Iowa. The first compartments of existing compartment mills were operated in closed circuit with rake classifiers, the second compartments with rake-bowl classifiers. In 1938 Universal Atlas Cement Co. installed at Mill No. 8 a complete closed circuit wet grinding system using 350-hp preliminary mills followed by similar secondary mills. An identical grinding system was installed at Mill No. 2 in 1944, and similar installations have been made elsewhere. In 1950 the Missouri Portland Cement Co. at St. Louis installed single-stage closed circuit grinding, using a 1000-hp ball mill in closed circuit with a rake-bowl classifier.

Compared with open circuit grinding, the gain in efficiency in all these cases was phenomenal—75 to 125 pct in output per installed mill horsepower, or 43 to 55 pct lower mill power consumption per unit of output. Ball wear per ton of product, which is considerably higher for wet grinding than for dry, was greatly reduced.

Generally there has not been much trouble using large diameter thickeners to bring the 80 pct moisture slurry down to normal kiln feed moisture. In some instances, however, there were difficulties, particularly with limestones containing colloidal impurities, or with fine colloidal clay or shale used as the argillaceous component of the raw materials.

Although they are seldom serious, these occasional difficulties have slowed the trend toward wet closed circuit grinding in the cement industry. Some companies have installed wet, open circuit compartment mills, with various forms of screens to control the tramp oversize, preferring the lower efficiency to the unknown difficulties of slurry thickening.

Combined Raw and Finish Grinding Departments: To save labor and simplify the electrical system, it is highly desirable to have all grinding mills, both raw and clinker, arranged in one building. A single crane, for servicing and repairing all the mills, is a great advantage. It is also convenient to set the mills in a line high enough for a truck to pass beneath them from one end of the mill room to the other. The service crane and truck passageway greatly facilitate handling manhole covers, repair parts, and ball charges.

Thermal Drying of Fine Coal

by G. L. Judy and H. L. Washburn

DURING recent years thermal drying of fine coal has increased at a tremendous rate, but very little discussion of the several excellent units has found its way into print. This is particularly true for operating and maintenance costs, and the operator who is faced with his first fine coal drying installation must depend chiefly on hearsay when trying to determine what his drying costs will be.

This article presents the maintenance and operating costs for the seven drying units now installed in preparation plants of Pittsburgh Consolidation Coal Co. and lists other cost information that should be obtained before any particular drying unit is selected.

Plant Description: The fine coal drying section of Plant A consists of four Flash driers operating on two furnaces equipped with spreader stokers. Plant B has one Flash drier operating (together with a screen-type coarse coal drier) on one furnace equipped with a spreader stoker. Plant C has two Cascade-type driers operating on a single furnace with a spreader stoker.

Plant Costs: Tables I, II, and III present the cost of operating these units as reported by the individual plants. As can be seen, the labor rates vary, depending on the amount of overtime paid and the time interval the cost figures cover. There is considerable variation in thermal units used and in the value given to the thermal units at the particular mine. There is some difference in power costs at a given location and, finally, the power costs of conveyors leading to the drying plants were included in some cases. Maintenance supply costs are simply a tabulation of the cost of all supplies used on the driers, furnaces, and stokers. Depreciation is not included, and the tables omit items such as supervision, plant cleanup, oiling, general maintenance, taxes, and insurance.

These tables in themselves do not offer a true comparison of costs. This information must be reduced to some common basis. But what is this common basis to be? A mine operator must calculate his overall mine and preparation plant costs in terms of tons of clean coal sold. Unit costs on pieces of equipment within a given plant are correctly calculated on the same basis. However, when unit costs are compared between one plant and another, some thought must be given to unit design. This reasoning is similar to the old argument concerning solid bowl vs basket-type centrifuges—the units themselves are not directly comparable without their supporting equipment. So it is with thermal driers. Except in rare cases of low entering moisture content, driers are sized on the basis of water that must be evaporated, while the wear each unit gets de-

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Table I. Flash Drying Costs, Including Furnaces, for 1955 at Preparation Plant A of Pittsburgh Consolidation Coal Co.

Item	Amount, $	Cost Per Ton of Coal (at 0.0 Pct Surface Moisture) Dried by Equipment, $
Operating Labor (Average rate = $2.525)	10,705.01	0.0103
Supply	—	—
Maintenance Labor (Average rate = $3.641)	26,426.83	0.0255
Supply	26,382.78	0.0255
Fuel @ $0.75 per ton (9,000 Btu)	22,132.80	0.0214
*Power (1056.4 connected horsepower)	30,294.06	0.0292
Total Cost	115,941.48	0.1119
Size dried		⅜ x 0 in.
Tons @ 0.0 pct surface moisture		1,036,408
Operation of driers, hr		4,240
Tons per hour dried (@ 0.0 pct surface moisture)		244.4
Surface moisture input		8.5 pct
Surface moisture output		2.5 pct
Seam		Pittsburgh
Inherent moisture		1.9 to 2.0 pct

* Actual connected horsepower on driers and furnaces = 779 hp. Power costs calculated from following information: $161,750.87 power cost for 5640 connected horsepower for 4240 operating hours.

Table II. Flash Drying Costs, Including Furnaces, for 1955 at Preparation Plant B of Pittsburgh Consolidation Coal Co.

Item	Amount, $	Cost Per Ton of Coal (at 0.0 Pct Surface Moisture) Dried by Equipment, $
Operating Labor (Average rate = $2.967)	2,699.10	0.0271
Supply	—	—
Maintenance Labor (Average rate = $3.531)	1,733.79	0.0174
Supply	553.59	0.0056
Fuel @ $3.02 per ton	2,932.42	0.0294
*Power (209 connected horsepower)	2,918.30	0.0293
Total Cost	10,837.29	0.1088
Size dried		3/16 x 0 in.
Tons @ 0.0 pct surface moisture		99,695
Operation of driers, hr		1815.50
Tons per hour dried (@ 0.0 pct surface moisture)		54.9
Surface moisture input		7.5 pct
Surface moisture output		2.5 pct
Seam		Pittsburgh
Inherent moisture		1.9 to 2.0 pct

* Actual connected horsepower on driers and furnaces = 192 hp. Power costs calculated from following information: $0.0103 per kw-hr and 1815.50 operating hours.

Table IV. Comparative Costs of Thermal Drying Units

Item	Preparation Plant A			Preparation Plant B			Preparation Plant C		
	Hours Per Ton of 0.0 Pct S.M. Coal Dried	Cost Per Ton of 0.0 Pct S.M. Coal Dried, $	Cost Per Ton of Water Evaporated, $	Hours Per Ton of 0.0 Pct S.M. Coal Dried	Cost Per Ton of 0.0 Pct S.M. Coal Dried, $	Cost Per Ton of Water Evaporated, $	Hours Per Ton of 0.0 Pct S.M. Coal Dried	Cost Per Ton of 0.0 Pct S.M. Coal Dried, $	Cost Per Ton of Water Evaporated, $
Operating Labor (rate $2.818)	0.004091	0.0115	0.171	0.009125	0.0257	0.463	0.009841	0.0277	0.305
Maintenance Labor (rate $2.986)	0.007003	0.0209	0.311	0.004925	0.0147	0.265	0.002253	0.0067	0.074
Supplies		0.0255	0.379		0.0056	0.101		0.0171	0.188
Subtotal	0.011094	0.0579	0.861	0.014050	0.0460	0.829	0.012094	0.0515	0.567
Fuel (at 2200 Btu per lb of water evaporated and 16¢ per MM Btu		0.0473	0.703		0.0390	0.704		0.0639	0.705
Power (at $0.007 per connected horsepower)		0.0223 (779 conn. hp)	0.331 (779 conn. hp)		0.0245 (192 conn. hp)	0.442 (192 conn. hp)		0.0233 (366 conn. hp)	0.257 (366 conn. hp)
Total cost		0.1275	1.895		0.1095	1.975		0.1387	1.529
Composite of Following Costs in Proportion to Their Tonnage									
1. Cost per ton of 0.0 pct S.M. coal dried 2. Cost per ton of water evaporated.	0.2388				0.2074			0.2544	

pends chiefly on the amount of coal handled. Therefore, when drying units in different plants are compared, the cost per unit weight of water evaporated must be calculated as well as the cost per unit weight of dry coal handled.

The writers have chosen a set of conditions and made these calculations, presented in Table IV. The labor rate happens to be that in effect at one of the Pittsburgh Consolidation Coal Co. mines. It has been assumed that an efficient drying plant should be able to operate on 2200 Btu per lb of water evaporated and that those thermal units have a value of $0.16 per million. Average cost of power per connected horsepower-hour for the three plants was selected as representative for this calculation. Table IV reveals that on the basis of cost per ton of 0.0 pct surface moisture coal Plant C has the highest cost and Plant B the lowest. On the basis of cost per ton of water evaporated the standings are reversed—Plant C has the lowest cost and Plant B the highest. It is interesting to compare the weighted composite of these two cost figures for the three plants.

	Plant A	Plant B	Plant C
Weighted composite of the cost per ton of 0 pct surface moisture coal dried and the cost per ton of water evaporated.	$0.2388	$0.2074	$0.2544

Inaccuracies in the tabulation of costs presented here could result in the difference between the high and low values shown above, and the writers believe that if these two types of driers were fed the same product in the same plant, maintenance and operating costs would be about the same. It must be remembered that these costs do not include depreciation and general plant overhead, and these are significant items. It must also be emphasized that cost of drying is not a fixed value that will stand up in all operating conditions and locations. It is a function of individual conditions. If the information presented in this article is to be of value the following steps should be taken:

1) A calculation (such as Table IV) should be made on the basis of labor, fuel, and power costs at a particular mine.

2) A depreciation cost for the installation must be determined. Since driers of different design are not interchangeable on a capacity basis, there can be significant differences in capital costs between installations.

3) The plant layout should be studied to determine if the use of other types of driers will result in an overall saving in plant structure and conveyors.

4) Finally, the purchaser should determine, to his own satisfaction, that the drying units offered will

Table III. Cascade Drying Costs from May 1953 to November 1956 at Preparation Plant C of Pittsburgh Consolidation Coal Co.

Item	Amount, $	Cost Per Ton of Coal (at 0.0 Pct Surface Moisture) Dried by Equipment, $
Operating Labor (Average rate = $2.480)	36,683.00	0.0244
Supply	—	—
Maintenance Labor (Average rate = $2.648)	9,240.00	0.0062
Supply	25,593.00	0.0171
Fuel @ $3.85 per ton	58,958.00	0.0393
*Power (366 connected horsepower)	31,702.00	0.0211
Total Cost	162,176.00	0.1081
Size dried		3/8 x 0 in.
Tons @ 0.0 pct surface moisture		1,500,812
Operation of driers, hr		13,660
Tons per hour dried (@ 0.0 pct surface moisture)		109.9
Surface moisture input		10 pct
Surface moisture output		2.0 pct
Seam		Pittsburgh
Inherent moisture		1.0 pct

* Power calculated on basis of horsepower actually connected to driers and furnaces, as follows: 85 pct efficiency—13,660 operating hours—0.01¢ per kw-hr.

evaporate the guaranteed amount of water under the operating conditions set out in the guarantee.

From a cost standpoint, items 2 and 3 are particularly important. Experience has shown that when all other factors are equal the decision as to which drying unit to use is often determined by unit and structural costs alone.

Although there are many factors other than costs to be considered before a fine coal drying unit can be selected that is right for the particular job, this information will provide a starting point.

Selection of equipment cannot be made without a thorough cost analysis. The proper piece of equipment for an individual installation cannot always be the cheapest unit, but a knowledge of the price being paid for certain advantages is essential.

Technical Note

Danger Period in Coal Mines Following a Low Pressure Passage

by Charles Barron McIntosh

BECAUSE of the well known relationship between a low atmospheric pressure and increased amounts of methane in coal mines, attempts have been made to find associations between low pressures and coal mine explosions.

An early study by F. Able disclaimed this relationship. He found that half the explosions took place during a rising barometer when methane content in a ventilated mine would be decreasing. A more recent study by C. L. Hosler revealed a close relationship between anthracite mine explosions and low atmospheric pressure. Bituminous coal mine explosions, where coal dust as well as methane can be exploded, were found to be more closely associated with a rising pressure following a low barometer than with the falling or low pressure period.

Emphasis on rising pressure periods led the present writer to investigate the effect of the post-frontal or post-cyclonic weather on coal dust.

Dust samples of uniform mesh were exposed in an Illinois mine. Analyses of the dust taken before and after the passage of cold fronts and their accompanying low pressure troughs indicated an increase in coal dust moisture prior to a frontal passage and loss of moisture from coal dust following the time of lowest pressure. Thus the warm, moist air (Tropical Maritime) found over the mine during the falling pressure period was cooled in the mine and moisture added to the dust. Cold, dry air (Polar Continental) accompanies the rising barometer following a frontal passage and the air, being heated in the mine, subtracted moisture from the coal dust. A similar but seasonal phenomenon occurs from summer to winter, the dry dust period in winter being recognized as the more dangerous period.

The question arose as to how the short period or daily changes in moisture content of coal dust compared with the seasonal changes. Temperature and humidity statistics were obtained from the surface and mine entry for the months of July and January. The gain and loss of moisture to the mine were calculated for these two months as were the gain and loss associated with mT (Tropical Maritime) and cP (Polar Continental) air masses as observed in the investigations. Temperature and humidity statistics taken from the two air masses represent a time span of only 34 hr. The resultant changes as presented in the accompanying table verify the supposition that short period changes in coal mine moisture may attain the magnitude of recognized dangerous seasonal changes. Statistics for the mT air and the July air have been placed next each other and those for cP and January air together for easier comparison of the two warm and two cold air units. The last column readily reveals that there is little difference in the amount of moisture being deposited in the mine from the mT air and the July average air. A similar conclusion is reached regarding the loss of moisture from the mine when it is affected by the cP air mass or the average January air.

Comparison of Seasonal and Air Mass Moisture Variations

Type of Air	Temperature, °F	Relative Humidity, Pct	Aqueous Vapor in Grains per Cubic Foot of Air	Gain or Loss to Mine in Grains per Cubic Foot	Gain or Loss to Mine in Gallons per Day
cP air mass	32.6	75	1.621	−4.250	−10,468
January average	29.5	73	1.382	−4.489	−11,059
mT air mass	71.6	82	6.887	+1.016	+2,506
July average	77.5	66	6.678	+0.807	+1,987

This comparison would be particularly true from late fall through early spring when strongly contrasting air mass changes are a common occurrence.

The invasion of cold, dry air following a cold front thus produces a drying period that could well receive more attention as a danger period for coal dust explosions. This drying period would compare with Able's explosion period during a rising barometer. The drying period is also pertinent in Hosler's findings, where it was noted that the average explosion day (for bituminous coal mines) followed the day of lowest atmospheric pressure. Moreover, the author's investigation of some 365 coal mine explosions also indicates that this post-frontal period should receive special attention in connection with mine safety measures.

Two comparatively recent major explosions in Illinois (Centralia and West Frankfort) occurred after the mines had been under the influence of cold-dry air for 25 to 30 hr following a deep low pressure passage. Dust was the important explosive element in both these explosions.

C. B. McINTOSH is with the Department of Geography, Eastern Illinois State University, Charleston, Ill.

TN 437F. Manuscript, June 28, 1957.

AIME OFFICERS:
PRESIDENT—GROVER J. HOLT
PAST-PRESIDENT—C. E. REISTLE, JR.
PRESIDENT-ELECT—A. B. KINZEL
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News of . . . Society Institute Profession

Coal-Energy Reservoir of North America Theme of Canada Conference

One of the social highlights of the AIME-CIM-ASME Joint Solid Fuels Conference promises to be the Banquet which will be held Thursday evening, October 11. Carroll Hardy will present the Percy Nicholls Award for 1957 at the banquet over which T. S. Spicer, professor of fuel technology at Pennsylvania State University, will be presiding.

Other social events planned are authors' breakfasts each morning and a business luncheon on Thurs-

(Continued on page 1154)

Cruise Planned for Post Annual Meeting Trip

A view of Pompano Beach, located on Bermuda's south shore. Bermuda will be the first of two stops on the Post Annual Meeting cruise.

Traditional part of the preparations for the AIME Annual Meeting includes plans for a post-convention trip—for relaxation only. This year anyone feeling the need for a holiday after the rigors of the meeting can bask in the sun and enjoy a leisurely triangle cruise—New York to Bermuda to Nassau and back to New York.

Arrangements have been made for space on the *Queen of Bermuda* for any members attending the 1958 Annual Meeting. For detailed information, contact Leon V. Arnold, 33 Washington Square West, New York, 11, N. Y.

The ship will depart from New York on February 21 and return on March 1. No hotel accommodations will be necessary at Bermuda or Nassau.

Time in ports will be sufficient to shop and see the major attractions on the islands.

Early requests will be accorded the choicest available space aboard ship.

Miners Will Travel To Tampa, Fla., for Five Day October Meeting

Final details are being completed by the committee and travel arrangements made by members of the Society of Mining Engineers of AIME who will be traveling to Tampa in October. This Florida city on October 15 to 18 will be the scene of the Southeastern States Mining Conference and first annual meeting of the Society of Mining Engineers and headquarters will be the Hillsboro and Tampa Terrace Hotels.

SME President Elmer A. Jones and AIME President-Elect Augustus B. Kinzel will be featured guests at the conference.

Some of the social highlights of the meeting are expected to be the welcoming luncheon and cocktail party on October 15, the banquet on October 16, and the deep sea fishing trip on October 19. Wives of those attending the meeting will be able to enjoy some of Florida's favorite attractions for visitors—Cypress Gardens and the Gulf coast beaches.

Several field trips are planned for October 17 and 18. On Thursday the phosphate field trips will offer opportunities to visit the Noralyn Mine, International Minerals & Chemical Corp.; the triple superphosphate plant, Davison Chemical Co.; or the phosphorus plant, American Agricultural Chemical Co. Alternate visits are planned for Friday to Florida Rock Products Inc., the Diamond Hill Mine, or Lehigh Portland Cement Co.'s plant.

(Continued on page 1158)

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The Society of Mining Engineers has announced that Charles E. Lawall, vice president, Chesapeake & Ohio Railway Co., Cleveland, has been nominated to serve as 1958 Eastern Vice President of the Society, and if elected, will take office in February 1958.

Broad Range of Papers Planned for Rocky Mountain Conference

The Annual Rocky Mountain Minerals Conference promises to be a combination of interesting technical sessions and social events. For news of the latter see August MINING ENGINEERING, and for the full tentative program see page 1155 of this issue.

The Cosmopolitan Hotel in Denver has been chosen as the headquarters of this fourth annual conference to be held from Oct. 30 to Nov. 1, 1957.

Mining Papers

The papers scheduled for the technical sessions cover a broad range of interests. One of those which will be of interest to mining men in general will be on the recent developments and exploration activities of the Cerro de Pasco Corp. This paper will be presented by Frank N. Spencer, Jr., who is the resident mining engineer at the New York office of the company.

Ground subsidence has always been a problem among mine operators and has received attention for many years. Richard M. Stewart, assistant to the director, mining research, The Anaconda Co., has prepared a paper giving a great deal of information on the subject of ground support with hydraulic backfill, a fast, inexpensive means of providing permanent ground support in an active stope.

William C. Campbell will deal with mining operations in his paper on shaft sinking at Homestake, where sinking has been in progress at four shafts in the Homestake Mine during the past five years. Various methods have been used by Homestake in the different shafts with some variation in equipment. Mr. Campbell is assistant mining superintendent at the Homestake Mine, Lead, S. D.

For the mining operator, ground water can be of intense interest and a paper on this subject has been prepared by Thad G. McLaughlin who, since 1951, has been the district geologist in charge of State-Federal ground water studies in Colorado.

Max E. Kofford, chief geologist of Western Gold and Uranium-Golden Crown Mining Co. has prepared a paper of general interest to mine operators which describes Orphan Mine.

Uranium

Uranium mining has for some years received a great deal of attention and with the intensive search for this mineral, sampling methods are of importance. A paper by John Q. St. Clair, consulting mining geologist, presents information on channel sampling with diamond and abrasive wheels, a method first developed on the Witwatersrand gold field in South Africa.

Harrison A. Schmitt, consulting geologist, will discuss copper in his paper on the Copper Province of the Southwest which includes Arizona and the neighboring parts of New Mexico and Sonora. This area produces 55 pct of the U. S. production or about 15 pct of the world's copper.

Other papers of general interest will be presented and included in the final program, available at the meeting. A symposium on metallurgy and panels on uranium mining and geology have been scheduled. The uranium panels will probably be made up of eight or ten men who are recognized as experts.

Coal Conference

(Continued from page 1153)

day at which C. Ouellet, dean of the Faculty of Science, Laval University, will be the speaker.

Some of the technical papers to be presented will be available in preprint form. The ASME papers so available will be listed by number in the program distributed at the meeting. Copies of these can be obtained from the ASME Order Dept., 29 West 39th St., New York 18, N. Y. Prices are 25¢ each to members of ASME and 50¢ to nonmembers. Papers must be ordered by their numbers. The final listing of available technical papers will be found in the issue of *Mechanical Engineering* containing an account of the conference.

AIME Joint Solid Fuels Conference papers submitted for publication will be considered for the Mining Transactions section and the Feature section of MINING ENGINEERING and if accepted will be published accordingly in MINING ENGINEERING. A limited number of these papers may be obtained from the Coal Division Secretary, D. R. Mitchell.

CIM papers submitted for publication will be considered for the *Canadian Mining and Metallurgical Bulletin*. For separate copies write to CIM headquarters, 906 Drummond Building, 117 St. Catherine St., W., Montreal 2, Canada.

Members of the Coordinating Committee are: E. R. Mitchell, ASME, principal coordinator; A. Ignatieff, CIM, AIME; C. E. Baltzer; C. Gerow, CIM; Carl C. Kamm, CIM; E. Larochelle; G. Letendre, CIM; C. L. O'Brian, CIM; E. Swartzman, CIM; and A. A. Swinnerton, EIC, CIM. Biographies of some of the conference committee appeared in the September issue of MINING ENGINEERING on page 1016.

C. E. Baltzer is chairman of the publicity committee and is assisted by William Bradbury. The technical session and reproduction of papers committee is under the chairmanship of E. Swartzman. Technical papers chairman is H. B. Charmbury. Program chairman for ASME is L. F. Deming and for AIME, D. R. Mitchell. G. Letendre is in charge of entertainment.

Members of the Percy Nicholls Award committee are: Carroll Hardy, Elmer Kaiser, Edward Steidle, W. E. Reaser, and Harold B. Wickey. The ladies' activities committee includes: Mrs. G. Letendre, chairman; Mrs. C. Gerow; Mrs. A. Ignatieff; Mrs. E. R. Mitchell; Mrs. C. L. O'Brian; Mrs. T. S. Spicer; Mrs. E. Swartzman; and Mrs. R. H. Taschereau.

The St. Lawrence Seaway and Power Project, when completed, will have an effect on the use of coal for producing electricity. The Seaway will effect the marketing of coal and this subject will be discussed by John R. Firth, M. A. Hanna Co., at one of the technical sessions during the Solid Fuels Conference. At another session, conferees will catch up with the Seaway progress via a color film *The Eighth Sea* to be presented by H. V. Tidman, Hewitt Equipment Ltd. The view above was taken a year ago between Iroquois and Cornwall, Ont., a point which, in 1955, had been apple orchards. When finished the area will become one of the major canal and lock systems of the Seaway.

Rocky Mountain Minerals Conference Program

WEDNESDAY, OCTOBER 30, AM

Registration

9:00 to 12:00 Cosmopolitan Hotel Mezzanine

Ladies' Coffee

9:00 to 12:00 Cosmopolitan Hotel Club Room

President's Luncheon

12:00 to 2:00 Cosmopolitan Hotel Silver Glade
Speaker: *Grover J. Holt*, AIME President

WEDNESDAY, OCTOBER 30, PM

Registration

2:00 to 4:00 Cosmopolitan Hotel Mezzanine

Ladies' Coffee

2:00 to 4:00 Cosmopolitan Hotel Club Room

Technical Session

2:00 to 5:00

Milling Practice on the Mesabi Range: *Earl C. Herkenhoff*, Utah Construction Co.

Shaft Sinking at Homestake: *Wm. C. Campbell*, Assistant Mine Superintendent

Sampling at Rio de Oro: *John G. St. Claire*, Consulting Geologist

Ground Water in Colorado: *Thad G. McLaughlin*, USGS

Mining Companies and Suppliers Cocktail Party

6:00 to 7:00 University Club

THURSDAY, OCTOBER 31, AM

Western Breakfast

7:30 to 9:00 Cosmopolitan Hotel

Registration

9:00 to 12:00 Cosmopolitan Hotel Mezzanine

Technical Session

9:00 to 12:00

The Metallurgy of Columbium and Tantalum: *Sherman May*, USBM

Tunneling: *Theodore F. Adams*, Blue River Constructors

Recent Developments, Exploration Activities of the Cerro de Pasco Corp.: *F. N. Spencer, Jr.*, Cerro de Pasco Corp.

Magma's Limestone Replacement Orebody: *Russell N. Webster*, Magma Copper Co.

Division Luncheons

12:00 to 1:30

Ladies' Luncheon and Book Review

12:00 Columbine Country Club
Review by *Mollie Lee Beresford*
Busses leave Cosmopolitan Hotel at 11:30 am.

THURSDAY, OCTOBER 31, PM

Technical Session

2:00 to 5:00

Pima Milling Operations: *G. A. Komadina*, Pima Mining Co.

Hydraulic Backfilling: *Richard M. Stewart*, The Anaconda Co.

A Survey of Electromagnetic Prospecting Methods: *George R. Rogers*, Bear Creek Mining Co.

Status of Copper Mining in Southern Arizona: *Harrison A. Schmitt*, Mining Geologist

FRIDAY, NOVEMBER 1, AM

Ladies' Coffee

9:00 to 12:00 Cosmopolitan Hotel Club Room

Technical Session

9:00 to 12:00

Milling Practice of American Chrome Co.: *Wm. H. Hisle*, American Chrome Co.

Recent Trends in Mining Machinery Development: *C. C. Frush*, Colorado School of Mines

The Orphan Mine—Grand Canyon, Ariz.: *Max E. Kofford*, Golden Crown Mining Co.

Luncheon for Alumni Groups

FRIDAY, NOVEMBER 1, PM

Technical Session

1:30 to 4:30

Panel: Uranium Mining and Geology
Panel: Uranium Metallurgy

Informal Dinner-Dance

7:00 Cosmopolitan Hotel Silver Glade

SATURDAY, NOVEMBER 2

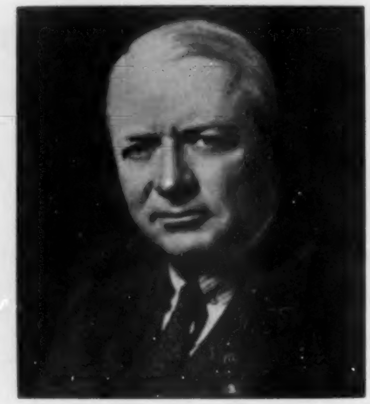
Brewery Tour

9:45 am Coors Brewery Golden, Colo.
Bus trip leaves Denver at 9:45 am. Box Lunch and refreshments enroute.

Football Game

Colorado University vs Missouri University

Scott Turner, Mining Engineer, To Receive Hoover Medal for 1957



In recognition of many years of outstanding public service in many capacities, Scott Turner, of New York, former director of the U. S. Bureau of Mines, has been named by the four major national engineering organizations to receive their jointly sponsored Hoover Medal for 1957.

Named after former President Herbert Hoover, who was its first recipient in 1930, it is awarded "by engineers to a fellow engineer for distinguished public service." It is one of the most distinguished honors bestowed by the profession. The sponsors are AIME, ASCE, ASME, and the AIEE.

Presentation will be in New York next February at the Annual Meeting of AIME, of which Mr. Hooover was President in 1920 and which Mr. Turner also headed as president.

A statement by the medal board of award relates that it was created "to honor engineers whose pre-eminent services have advanced the well-being of mankind and whose talents have been devoted to the development of a richer and more enduring civilization" and that "the engineering societies award the Hoover Medal in recognition and appreciation of those principles and ideals of civic obligation and of public service exemplified by the life and work of Herbert Hoover."

Mr. Turner's career, both as a mining engineer and as the head or member of major Government missions, has taken him to many countries and his achievements in both fields and in education have been hailed on many occasions. Some of his enterprises involved extremely hazardous situations and exacting diplomatic skill. He is the author of some 50 technical papers and has been the recipient of many formal honors from professional societies and universities.

Born in Lansing, Mich., Scott Turner received an A.B. in science from the University of Michigan, which later conferred upon him the honorary degree of doctor of engineering. He received the degrees of B.S. and E.M. from Michigan College of Mines, which later, as the Michigan College of Mining and Technology, awarded him the honorary degree of doctor of engineering. He also holds the honorary degree of doctor of science from Colorado School of Mines and from Kenyon College.

Mr. Turner's early engineering years were occupied with a variety of activities in several Western states and in Alaska and by studies of properties in Northern Africa, Spain, Norway, Sweden, and other areas. Named in 1911 as general manager of a vast coal mining project in the far north, he had the responsibility of maintaining American sovereignty over some 600 square miles of an unclaimed, uninhabited *no man's land* 700 miles from the North Pole. Mr. Turner wrote and administered laws; organized measures for the health, safety, food, and employment of 600 men; designed and built docks, tramways, industrial railways, electric power plants; opened and operated mines producing high grade semibituminous coal; created an industrial town; and in many other ways met, mastered, and maintained hazardous and trying conditions in a country where permafrost was 2000 ft thick.

Not the least of the exacting demands on Mr. Turner's administration of affairs were during the eight months when the Spotsbergen route was ice-blocked. Exceptional diplomatic tact was required in the ejection of trespassing groups sent by various European countries, particularly Russia, while at the same time maintaining working relations with envious European governments.

When World War I brought new dangers to a normally difficult situation, the properties were optioned to Russia. En route from Boston to Petrograd to complete the sale, Mr. Turner was injured in the disastrous torpedoing of the Lusitania, on which he was a passenger. He was hospitalized in London. Russia's chaotic turn of events turned Mr. Turner toward Norway, where he sold the properties to a syndicate.

A few weeks later, he began for a group of London bankers an important two-year study of mines in South America. He returned to the U. S. to receive a commission as lieutenant, senior grade, in the Navy. He served under the chief of the Bureau of Engineering. After the war, Mr Turner was technical head of the Mining Corp. of Canada for seven years, helping in the development and operation of silver and copper mines in the Dominion and properties in other parts of the world. After 17 years in foreign countries, Mr. Turner returned to become director of the Bureau of Mines under Secretary of Com-

(Continued on page 1158)

ROCK IN THE BOX

News of M.G.G. Division

We have recently received a well deserved complaint for using the initials MGG in the above caption. Let's clarify the situation once again and inform our membership that if your major interests are in mining, geology, or geophysics, you belong in the MGG Division. Oops—we used those initials again. We hope to get away from this problem and would like now to present the solution. Please study the proposed new Bylaws of our Division that are presented below. These suggestions came from the Division's Executive Committee.

First—We suggest a change in the name to the *Mining and Exploration Division* and invite the geochemists to join our group.

Second—We revised the slate of officers to represent the entire Division rather than particular functions in our group.

Third—The Bylaws provide Unit Committees that accommodate activities of the special interests in our group.

Fourth—This proposal eliminates duplication of effort in program, publications, and membership activities. Your Executive Committee hopes that these new rules will eliminate the old feeling of breaking our Division into Subdivisions of special interest, and will tend to strengthen the overall organization.

These Bylaws were approved by a majority of the MGG Executive Committee and are presented here to the membership for their information and study. Please vote and indicate your desire in the acceptance of these Bylaws. Fill in the ballot, **right**, and send it in by October 31 to Glen A. Burt.

We trust that these suggested changes will be approved by our members and the AIME Board of Directors. If so, they will become effective at the time of our Annual Meeting, February 1958, in New York.

C. L. Wilson

Article I

Name and Object

Section 1. This Division of the Society of Mining Engineers of the Americal Institute of Mining, Metallurgical, and Petroleum Engineers shall be known as the Mining and Exploration Division.

Section 2. The objective of the Mining and Exploration Division shall be to furnish a medium of cooperation between those directly engaged in metal mining and those engaged in the technologies of mineral exploration. To further the objective toward a common goal and advance these branches of the mining industry, this Division will promote and publish papers, arrange meetings and programs, and encourage education on any subject related to these phases of the industry.

Article II

Members

Section 1. Any member of the American Institute of Mining, Metallurgical, and Petroleum Engineers and the Society of Mining Engineers may become a member of this Division by indicating to the Society of Mining Engineers in writing his desire to do so.

Article III

Officers

Section 1. *Division Officers.* There shall be a Chairman, Assistant Chairman, three Vice Chairmen, and a Secretary equitably representing the membership of the various fields of interest within the Division. The Division offices should be rotated to represent the various fields of interest as qualified candidates become available.

Section 2. *Nomination and Election of officers.* The Nominating Committee shall report to the Chairman on or before May 15 the nominees for Chairman, Assistant Chairman, and three Vice Chairmen. In turn the Chairman shall arrange for the publication of the names submitted in the July issue of MINING ENGINEERING and shall also advise the Secretary of the Society of Mining Engineers of the nominations. Other nominations for office may be made and forwarded in writing to the Secretary of the Society up to August 15, for publication in the October isssue of MINING ENGINEERING. If such nominations are made, letter ballots will be prepared for return not later than November 1. If no other nominations are received the candidates nominated by the Committee will be considered elected and will take office at the Annual Meeting.

Article IV

Committees

Section 1. *Executive Committee.* The Executive Committee shall consist of the officers listed in **Article III**, *Section 1*, the Chairman of each of the Unit Committees, and the most recent Past-Chairman of the Division. For the transaction of business the presence of a quorum of not less than five (5) members shall be necessary. If there are less than the required quorum of five (5) present at the meeting, the meeting shall be held and the minutes circulated to the entire Committee for approval.

(Continued on page 1159)

Mining, Geology, and Geophysics Division Members ____________

VOTE NOW

Please signify your approval or disapproval of the proposed Bylaws by marking an *X* in the proper box below. **Sign and return this ballot** (before midnight, October 31) to: *Glen A. Burt, MGGD Secretary, 808 Kearns Building, Salt Lake City, Utah.*

I approve the Proposed Bylaws as published in the October 1957 issue of MINING ENGINEERING

☐ YES ☐ NO Signed:____________________

NOTE: Ballots will be invalid unless marked, signed, and postmarked no later than midnight, Oct. 31, 1957.

Scott Turner

(Continued from page 1156)

merce Hoover, continuing when Mr. Hoover became President. Mr. Turner served in Washington nine years. Later he became associated with various mining activities as executive or consultant.

Important government assignments took him to many areas. Among the agencies for whom Mr. Turner engaged in public services were the Federal Oil Conservation Board, National Research Council, Research Committee of Engineering Foundation, International Geological Congress, council and board of the World Power Conference, State Dept., Dept. of Justice, Secretary of the Treasury, and Secretary of War. Mr. Turner participated in international mining and world engineering congresses and other major world meetings.

Mr. Turner became a member of AIME in 1906 and President in 1932. He is a member of the Mining and Metallurgical Soc. of America, CIM, and is an honorary member of the American Zinc Inst. and of the Coal Mining Inst. of America. He served two terms as president of the American Inst. of Consulting Engineers.

He has engaged in many educational activities, including years as a director and member of the finance committee of the Belgian-American Educational Foundation. He is a member of the American Relief Assn. Mr. Turner was one of the original group of five men delegated to prepare Canons of Ethics in Engineering, part of the time being chairman of the committee. He has been a chairman of Student Guidance in the metropolitan area of New York.

Tampa Meeting

(Continued from page 1153)

Since the September issue—in which the technical program was published on pp. 1014 and 1015—went to press, some changes and additions have been made.

The paper *Some X-Ray Diffraction Studies of the Leached Zone Minerals* by Robert W. Gould has been transferred from the Tuesday morning, October 15, **Geology and Geophysics** session to a new session added on Tuesday afternoon. Inserted in its place at 10:30 am is *The Occurrence of Oil in Florida* by John D. Bates and Joseph E. Banks, Coastal Petroleum Co.

The Tuesday afternoon, October 15, **Minerals Beneficiation** session is to be a joint one with **Industrial Minerals.**

The paper *Economic Evaluation of an Industrial Mineral Project,* to be given at 2:30 pm, Tuesday, October 15, at the **Mineral Economics** session, was written by C. M. Edwards, International Minerals and Chemical Corp. C. E. Gable of the same company will present the paper for Mr. Edwards.

Two additional sessions for **Mineral Economics** and **Geology and Geophysics** have been scheduled for Tuesday and Wednesday afternoons, October 15 and 16. The program of these sessions follows:

TUESDAY, OCTOBER 15, PM

Geology and Geophysics

2:00—Sand Ilmenites of the Eastern United States: *Harry B. Cannon,* H. B. Cannon Assoc.

2:30—Some X-Ray Diffraction Studies of the Leached Zone Minerals: *Robert W. Gould,* American Cyanamid Co.

3:00—The Hawthorne and Alachua Formations of Alachua County, Florida: *E. C. Pirkle,* University of Florida.

3:30—Topographic, Geologic, and Hydrologic Factors Affecting Methods of Treatment and Disposal of Radioactive Wastes from Nuclear Reactors: Stanley O. Reichert, University of Florida.

WEDNESDAY, OCTOBER 16, PM

Mineral Economics

9:30—Productivity in American Industry: *Speaker* from U. S. Dept. of Labor.

10:00—The Consolidation and Expansion of Markets for Zinc: *J. L. Kimberley,* American Zinc Institute.

10:30—Broad Aspects of the Present Aluminum Industry: *Paul Brant,* Reynolds Metals Co.

11:00—State of the Economy: *Henry Toland,* Exchange National Bank.

The Wednesday morning, October 16, **Minerals Beneficiation-Industrial Minerals** session will be a **Minerals Beneficiation** session only.

The paper *Sand Ilmenites of the Eastern United States* by Harry B. Cannon has been transferred from the Wednesday afternoon, October 16, **Mining** session to the new **Geology and Geophysics** session on Tuesday afternoon. The papers originally listed for presentation at 3:00, 3:30, and 4:00 pm will now be presented at 2:30, 3:00, and 3:30 pm. The last two papers on the program for this session will deal with *Nerex*-liquid oxygen explosives, not Merex as originally listed.

Any other program changes will be listed in the final program, available at the meeting.

Proposed Bylaws of the Mining and Exploration Division of SME

(*Continued from page 1157*)

Section 2. *Unit Committees.* There shall be a minimum of five (5) permanent Unit Committees representing 1) Underground Mining, 2) Open Pit Mining, 3) Geology, 4) Geophysics, and 5) Geochemistry. Other Committees representing any phase of the industry may be created at the discretion of the Division Chairman, as the need arises. The Committee Chairmen shall be appointed by the Chairman of the Division upon the advice of those members interested in that particular phase of the industry. Should no selection be made by the members, the Chairman shall appoint the Committee Chairmen.

Section 3. *Nominating Committee.* The Nominating Committee shall consist of a minimum of eight (8) members, including the immediate Past-Chairman, the past Unit Committee Chairmen, and two (2) others to be selected by the Chairman of the Division.

Article V

Duties of Officers

Section 1a. The Chairman shall preside at the annual business meeting of the Division, which will take place at the time of the Society of Mining Engineers Annual Meeting. He shall call other meetings as required to transact the business of the Division. He shall be responsible for coordinating the programs for the meetings of the Division. He shall name the Chairmen of the Unit Committees, as set forth in **Article IV,** *Section 2.* He shall name the Secretary of the Division.

b. The Assistant Chairman shall keep in constant touch with all actions of the Chairman and shall act in his stead whenever the Chairman is unable to function or attend any meetings. He shall assist the Chairman as directed.

c. The Vice Chairmen shall be designated as follows: 1) Vice Chairman, Publications; 2) Vice Chairman, Program; 3) Vice Chairman, Membership. It shall be the duty of each Vice Chairman to coordinate the Unit Committees' activities under his classification. Each shall act as a liaison officer between the Division and the appropriate officer or committee of the Society of Mining Engineers. They shall at all times keep the Division Chairman and Assistant Chairman informed of their activities.

d. The Secretary of the Division will send out meeting notices, take the minutes of the Executive Committee or business meetings, follow instructions of the Chairman of the Division and Secretary of the Society of Mining Engineers, and perform such other duties which may be necessary for proper functioning of the Division.

Article VI

Duties of the Committees

a. The Executive Committee shall be charged with the responsibility of conducting the affairs of the Division in a businesslike manner. It shall have the authority to appoint a Chairman Pro Tem to act when any of the officers are unable to function.

b. The Unit Committee Chairmen shall appoint from members interested in their particular unit, Unit Committee members on 1) Publications, 2) Program, 3) Membership, plus 4) any other members for special investigations or projects that they may see fit to investigate. Each Unit Committee Chairman shall inform his committeemen of the Vice Chairman in charge of their specific activity and also inform the Vice Chairman of the member appointed. The Unit Committee Chairman may allow the committee member to report directly to the Vice Chairman or may act as an intermediary. In either case, the Unit Committee Chairman is responsible for proper functioning of his committee members. It shall also be the duty of the Unit Committee Chairman to canvass the members of his group and to report to the Division Chairman the name of the member selected to succeed him as Unit Committee Chairman. This shall be done at least two months in advance of the Annual Meeting.

c. The Nominating Committee shall meet to make their selections in time to comply with the deadline set in **Article III,** *Section 2.* Division members having candidates should submit the name of the candidate to the Committee Chairman prior to May 1 of each year. The Committee shall give due consideration to all names submitted.

Article VII

Funds

Funds received by or assigned to the Division shall be deposited with the Secretary of the Society of Mining Engineers at New York, or at any other place and under the responsibility of an officer of the Division as deemed necessary by the Executive Committee for the efficient operation of the Division. The Secretary or other responsible party shall submit a statement of receipts and disbursements to the Chairman of the Division in time for a report for the annual business meeting. Disbursements from Division funds may be made by the Secretary of the Society of Mining Engineers upon authorization of both the Division Chairman and the Division Secretary, for such purposes as have been authorized by the Executive Committee of the Division. Exceptions to this general rule can be made by order of the Executive Committee of the Division.

Article VIII

Meetings

Section 1. The Division shall meet at the same time and place as the Annual Meeting of the Society of Mining Engineers for the election of Division officers and for the transaction of any other business; and at such other times and places as may be determined by the Executive Committee. Notice of such a meeting must have been sent to the members of the Division through the regular mail, or must have been published in MINING ENGINEERING; to reach the members at least 20 days before the meeting.

Section 2. For the transaction of any business, the presence of a quorum of not less than twenty (20) members shall be necessary.

Section 3. At the annual business meeting of the Division the order of business shall be as follows: The meeting will be called to order by the retiring Chairman who will transact any necessary business and present any reports or call for any deemed necessary. He will then introduce the new officers and turn the meeting over to the new Chairman. The new Chairman will announce the names of the new Executive Committee, appoint a nominating committee, and transact such other business as necessary.

Article IX

Amendments

Proposals to amend these Bylaws shall be made in writing to the Executive Committee of the Division and signed by at least ten (10) members. They shall be considered by the Executive Committee and announced to the members through the columns of MINING ENGINEERING, together with any comments or amendments made by the Executive Committee thereon. They shall be voted upon at the Annual Meeting of the Division, or by letter ballot, as may be directed by the Executive Committee and are subject to approval of the Board of Directors of the Society of Mining Engineers and the Board of Directors of the American Institute of Mining, Metallurgical, and Petroleum Engineers.

Around the Sections

Colorado Plateau Section was one of the stops made by Augustus B. Kinzel, AIME President-Elect, during his tour of operations in the Four Corners area July 12 to 14. Undoubtedly uranium and its peacetime uses in atomic energy, the subject of Dr. Kinzel's talk to the section, was the topic for animated discussion by members of the Section in this uranium-producing area. Shown, left to right, are: Richard Philippone, AEC, chairman of the Section; J. F. Benton, Union Carbide Nuclear Co.; Roy E. O'Brien, AIME Field Secretary; and Dr. Kinzel.

• The **Colorado Plateau Section** held a meeting on the evening of July 13 at which those attending heard Augustus B. Kinzel, one of America's most distinguished scientists and AIME President-Elect, speak on the necessity of keeping up the present mining rate of uranium. Dr. Kinzel stressed the point that the mineral is in as great demand in this country today as it was several years ago. Nuclear energy, he said, is just now beginning to take a place beside other forms of power, such as coal and oil. He stressed that in time, and if economic conditions justify an installation using atomic energy, we will see such units. Dr. Kinzel has been doing research for Union Carbide Corp. for 29 years and now is vice president for research. During his years with Union Carbide, he has also served with various AEC installations. His views are highly respected in the industry.

• AIME President Grover J. Holt was the guest of honor at the **Pennsylvania-Anthracite Section** annual summer meeting. Held at the Irem Country Club, Dallas, Pa., on June 27, the day's activities were devoted to golf, swimming, and other seasonal past times. About 200 members and their wives attended the cocktail party and dinner which climaxed the meeting events. Mr. Holt was the featured speaker and his talk ranged from Institute affairs to work being done in the iron ore field. Paul Goddard, Carey, Baxter & Kennedy Inc., was chairman of arrangements and Robert Walsh, Mine Safety Appliances Co., was chairman of the cocktail party which preceded the dinner. Emerson Todd was chairman and Tom Weichel, assistant chairman of the golf committee. Cocktail glasses embossed with the Section's insignia were presented as favors to those attending the meeting.

After the dinner, a short business meeting was held. Section officers elected at that time include: Walter B. Petzold, chairman; Harold B. Wickey, vice chairman; Emerson Todd, secretary-treasurer; and Floyd S. Sanders, ex-officio. Executive committee members elected for one year are: Garfield Schnee, John S. Marshall, Charles S. Kuebler, Robert R. Walsh, and Earl W. Lamb. Elected for two-year terms were: Thomas C. Price, Wesely Stonebraker, Donald Markle, Sr., James H. Pierce, and Francis O. Case. John H. Wilson, George H. Lovell, William C. M. Butler, Jr., Edward T. Powell, and William H. More were elected for three-year terms.

• The **Adirondack Section** held a midsummer meeting on August 24 at the Tupper Lake Country Club. During the day members enjoyed the excellent golf facilities. A panel of five members from several different companies gave short talks on quality control problems peculiar to their industry. The members of the panel and their subjects were: Roger Tolosky, Republic Steel, Lyon Mt., *Sampling Ore and Concentrates, and Laboratory Work;* Dexter Hatch, National Lead Co., Tahawus, *Specific Gravity Analyses of Ores;* Frank Woodworth, Jr., Jones & Laughlin Steel Corp., Star Lake, *Determining Minimum Mineable*

(Continued on page 1162)

Grover J. Holt was the guest of honor and principal speaker at the dinner which climaxed the Pennsylvania-Anthracite Section annual summer meeting. Officers for 1957-1958 were elected at that time. Seated at the head table are, left to right: Walter B. Petzold, incoming Section chairman; Floyd S. Sanders, retiring chairman, who has been a Section officer for many years; Mr. Holt; Mrs. Sanders; and E. O. Kirkendall, AIME Secretary, who accompanied Mr. Holt on his Section visit.

INDUSTRIAL MINERALS DIVISION

NEWSLETTER

Dear Members of the
Industrial Minerals Division:

We hope to see many of you at the Southeastern States Mining Conference and first Annual Meeting of the Society of Mining Engineers to be held October 15 to 18 in Tampa, Fla. Industrial minerals are well represented on the program and we hope to have a fine turnout from our Division. In any case, your Newsletter this month is intended to bring you up to date on Divisional affairs which have occurred during late spring and summer.

Some of you may have attended the regional conferences in Reno and Portland during April or will attend the forthcoming meeting of the American Mining Congress in Salt Lake City during September, and know that industrial minerals and developments in that field are in the forefront of the discussion. Tom Gillingham, our Program Chairman, is keeping things going by tying together a well rounded group of papers in our area for the AIME Annual Meeting in New York, Feb. 16 to 20, 1958.

Nominations

The last Executive Committee meeting of your Division was held in New York on May 20, 1957. The following list of officers for 1958 was nominated by the Divisional Nominating Committee (R. B. Ladoo, Chairman) and approved by the Executive Committee:

Nominated for *Chairman* was Robert M. Grogan and for *Secretary-Treasurer,* John G. Broughton. Those nominated to represent different regions as *Vice Chairmen* were: T. E. Gillingham, Northeast; Charles E. Hunter, Southeast; John B. Patton, Mid-Continent; Henry G. Fisk, Rocky Mountain; Howard A. Ziebell, Pacific Northwest; Lauren A. Wright, Pacific Southwest; and Alex B. Jackson, Canadian.

Nominated to serve on the *Executive Committee* for 1957 to 1958 was A. O. Bartell with Pauline Moyd as the alternate. Nominated to serve from 1958 to 1961 were: Frank R. Hunter, David M. Larrabee, and G. Donald Emigh.

A. B. Cummins and R. M. Foose were nominated for the *Society of Mining Engineers Nominating Committee* and G. H. Chambers and R. C. Stephenson were nominated as alternates.

Donald R. Irving was nominated for the *SME Transactions Editorial Committee* and R. C. Stephenson for the *SME General Editorial Committee.*

The SME Education Committee has been expanded, requiring three members from our Division instead of one. In addition to Sanford Cole, who has previously been appointed, our representatives now are J. E. Allen, Oregon, and J. M. Parker, North Carolina. Also at the Executive meeting, a report was made by the Bylaws Committee. These will be announced to the members later in the year and voted on at the Annual Meeting or before.

Publications

J. L. Gillson, Editor of the new edition of the Seeley W. Mudd volume, *Industrial Minerals and Rocks,* has made an encouraging report on the progress of that much needed book. Three quarters of the chapters have been written, and he anticipates that the entire volume should go to press shortly after the first of the year.

As you know, plans had been made to publish a second edition of the IndMD Directory this year. However, the SME has decided to publish a directory of the whole Society next year as a supplement to MINING ENGINEERING. Ray Ladoo, our representative on the Executive Committee of SME, indicates that this directory will follow closely the form of our first edition and, thus, should be satisfactory for the use of Divisional members.

Industrial Minerals Notes

With his usual keen interest in new and unusual developments in our field, Ray Ladoo has called our attention to the following items, some of them from the less widely read technical and trade publications.

Silicate fiber—The Carborundum Co. has announced a new fibrous synthetic aluminum product capable of withstanding over 2000°F. This fiber is reported to be flexible enough to be used with asbestiform textile materials such as yarn, tape, and woven fabrics.

Pollucite—A new and perhaps commercial source of cesium has been reported from drill cores in the spodumene deposits at Bernic Lake, Manitoba. Montgary Exploration reports that five drill cores have shown important concentrations of pollucite with testing incomplete. Drilling carried on so far has shown that there are considerable tonnages of spodumene ore in the deposits.

Silica—Ultra-high purity silicon (known as hyperpure silicon) has become of great and increasing importance for the making of transistors and other semiconductor electrical devices. At least four companies are now in commercial production and are working on the problem. Impurities have been reduced in some grades to as low as one part per billion. Hyperpure silicon is made from silicon tetrachloride or tetraiodide.

Bentonite—A relatively new and important market for bentonite has developed as a binder for the pelletizing of very fine taconite ore concentrates.

Feldspar Clays—The formation of a new company to produce feldspar and high grade clays has been announced. This company is known as the Lawson-United Feldspar and Minerals Co. and is a combination of United Clay Mineral Corp. and the Lawson feldspar interests. They are rebuilding the United Feldspar Plant at Spruce Pine, N. C., which was partly destroyed by fire some years ago. United has been producing ball clays in Tennessee and Maryland, and kaolin in Georgia, South Carolina, and Florida.

Gypsum-Anhydrite—The August 1957 issue of *Industrial and Engineering Chemistry* contains an 11-pp. world review of the status of the process for making sulfuric acid and Portland cement from gypsum and anhydrite. Information is given on eight plants in England, France, Germany, Poland, and Austria.

Phosphate Rock-North Carolina—Major industrial minerals concerns have been exploring the possibilities of commercial production of phosphate rock over a large area in eastern North Carolina. It is reported that phosphate beds located at depths of 100 to 150 ft occur over wide areas.

J.G.B.

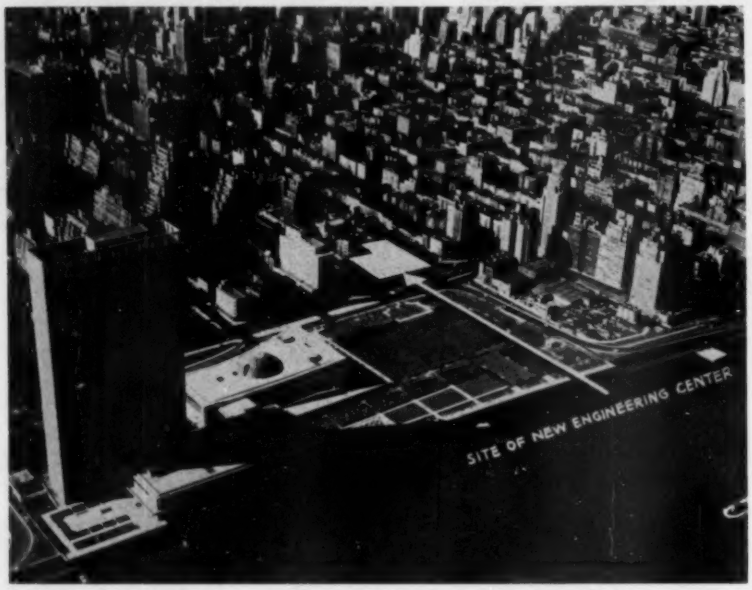
EJC-ECPD Program To Further Mutual Aims

The Engineers' Council for Professional Development and the Engineers Joint Council will hold a General Assembly at the Hotel Statler, New York, on Oct. 24 and 25, 1957. It will be the 25th annual meeting for ECPD and the fourth annual general assembly for EJC. The purpose of this joint program is to express the basic unity and complementary nature of the goals of the two groups.

The technical sessions will concentrate on the engineer's education and his role in various segments of society. The role of the two-year post high school curricula in technological education will be discussed as well as new dimensions in post-graduate education. There will be an evaluation of military service for the engineer and a panel discussion concerning programs designed to aid engineers to develop into management.

Luncheons will be given both days and Thursday evening, October 24, the Silver Anniversary Dinner of the ECPD will be held with C. E. Davies as the speaker.

Planners Prepare for New Engineering Center



A view of the site for the new Engineering Center from the East River looking west. The arrow indicates the site and the U. N. building can be seen to the left. Shreve, Lamb & Harmon Assoc. have been chosen as the architects for the new center which will be located on First Ave. Jaros, Baum & Bolles will be the mechanical consulting engineers and Seelye Stevenson Value & Knecht, the structural consulting engineers. For full information about the site, see MINING ENGINEERING, September issue, Page 969.

Mineral Industry Meetings

Oct. 3-5, Seventh Annual Exploration Drilling Symposium, University of Minnesota, Center for Continuation Study, Minneapolis.

Oct. 18, Illinois Mining Inst., 65th annual meeting, Hotel Abraham Lincoln, Springfield, Ill. **Oct. 19,** Illinois vs Minnesota, homecoming football game, Champaign, Ill.

Nov. 14-15, Third Annual Symposium on Mining Research, Missouri School of Mines and Metallurgy and USBM, Rolla, Mo.

Feb. 6-7, 1958, California Governor's Industrial Safety Conference, 8th annual statewide meeting, including Mineral Extraction section, Fairmont Hotel, San Francisco.

June 23-28, Third International Coal Preparation Congress, organized by the Centre d'Etudes et Recherches des Carbonnages de France, Brussels and Liege, Belgium.

NOTE

A limited number of copies of *Mining Branch Abstracts*, prepared for the AIME Annual Meeting in New Orleans, Feb. 24-28, 1957, are still available. Copies can be obtained by writing to John C. Fox, Society of Mining Engineers of AIME, 28 W. 39th St., New York 18, N. Y., and enclosing 50¢. Please indicate your Divisional interest at the time you order.

PERSONALS

Paul F. Karrow has been appointed to the geological staff of the Ontario Dept. of Mines.

Felix S. Dreyer and **Richard H. Lewin** have been elected to the board of directors of the Cerro de Pasco Corp.

N. T. Camicia, general manager of mines, Island Creek Coal Co., Huntington, W. Va., has been elected vice president and general manager of operations.

Arthur W. Storm has been appointed chief engineer of the new Surface Pelletizing Div., Surface Combustion Corp., Toledo.

John F. Myers has left Denver Equipment Co. to return to private consulting practice. He can be reached at 2 Putnam Hill, Greenwich, Conn.

H. Wm. Ahrenholz recently resigned from The New Jersey Zinc Co., Flat

(Continued on the next page)

Around the Sections

(Continued from page 1160)

Grades of Ore and Differentiating Magnetic and Nonmagnetic Ore; C. Kitts, International Talc Co., Gouverneur, *Mine Particle Testing;* and John R. Reilly, Jr., St. Joseph Lead Co., Balmat, *Quality Control in Sphalerite Flotation.* Panel moderator was John S. Cherry, Westinghouse Electric Corp. Dinner followed the panel discussion and Mr. Cherry was the speaker for the evening—his subject, statistical quality control techniques.

• **San Francisco Section's** initial meeting for 1957-1958 was held at the Engineer's Club on September 4. Speaker for the evening was W. R. Paxton who discussed *Brief Case for Business,* a timely topic in view of the present uncertainty in regard to the future of U. S. economic conditions. Standard Oil Co. of California provided the program for the meeting.

Caterpillar Tractor Co. has released *The Lowbowl Story,* an 8½-min 16-mm film, which illustrates the Lowbowl concept and principles in scraper design and which shows its advantages over other types. Introduced in 1955, the design has proved, through weighed loads on typical jobs, its loading characteristics and better traction. The film is available through your nearest Caterpillar dealer or in writing from the Advertising Div., Caterpillar Tractor Co., Peoria, Ill.

(Continued from preceding page)

Gap Mine, Treadway, Tenn., and is now professor of mining engineering at the University of Alabama, Tuscaloosa, Ala.

E. J. CARROLL

Edward J. Carroll has been named sales manager, Mining Tool Div., Kennametal Inc., Bedford, Pa. He had been assistant sales manager. Prior to joining Kennametal, Mr. Carroll was assistant superintendent, Robena Mine, Coal Div., U. S. Steel Corp.

Ernest M. Spokes has been promoted from associate to full professor of mining engineering at the University of Kentucky, Lexington, Ky.

Sherwin F. Kelly has spent the past year in geological and geophysical examinations of copper properties and uranium prospects in Cuba. His firm has just completed a geophysical examination of a nickel property in Southern California. Mr. Kelly is president of Sherwin F. Kelly Geophysical Services Inc., Amawalk, N. Y.

S. F. KELLY

A number of personal changes have been announced by Marion Power Shovel Co. **Adrien F. Busick, Jr.,** formerly vice president-engineering, has been promoted to executive vice president and general manager of the company, a division of Universal

Marion Corp. **William R. LeMasters,** who was recently named secretary-treasurer, has been elected to vice president. **Maurice V. Cornell** was named vice president for the sale of large machines, replacing **David E. Rizor** who retired. Mr. Cornell had been in the sales department. **Philip Kass** has been appointed executive staff assistant. He recently joined the company. Three changes have been announced in the engineering department: **Merle V. Lashey** has become manager of engineering; **Jack F. Weis** is now assistant chief engineer for both large and intermediate machines; and Robert W. Bergmann is now electrical engineer.

R. E. Tennery has been appointed general superintendent of Link-Belt Co.'s Los Angeles plant.

Ray H. Feierabend has been elected an assistant vice president of Freeport Sulphur Co., the firm has announced. He was recently transferred to the New Orleans office to become responsible for developmental work relating to Freeport's Grand Isle offshore project. Mr. Feierabend is MGG program chairman for the 1958 AIME Annual Meeting and was a member of the general committee for last year's Annual Meeting in New Orleans. A native of Little Rock, Ark., and a graduate, in 1942, of Columbia University, he joined Freeport as a mining engineer at its Hoskins Mound, Texas, mine.

R. H. FEIERABEND

Colorado School of Mines has named a three-man committee to direct the chemistry department in place of the usual department head. The committee is comprised of **Walter H. Dumke,** associate professor of chemistry; **John T. Williams,** assistant professor of chemistry; and **Robert A. Baxter,** professor and member of the chemistry faculty since 1922. **W. W. Howe,** former head of the department retired in June after teaching at the School for 28 years.

W. E. Field, formerly manager of the Basin-Jib Mines Ltd., Basin, Mont., is now with National Explorations Ltd., Uranium City, Sask., Canada, as manager.

Donald T. Delicate, formerly superintendent of mines at Homestake Mining Co., Utah Div., is now with Homestake-Sapin Partners, Grants, N. M., as mine superintendent.

Frank C. Pickard, previously with The American Metal Co. Ltd. as a consultant, is now with International Minerals & Chemical Corp. as project engineer in charge of construction for the company's Canadian potash project.

Dietrich Rontz is with Denver Air Machinery Co., Denver. He had been with Le Roi Div., Westinghouse Air Brake Co., Milwaukee.

Paul Novotny, formerly with Pacific Gas & Electric Co. as an engineering assistant, is now with Kennecott Copper Corp. as a junior engineer.

Hubert D. Keiser, chief of the U. S. Bureau of Mines, Branch of Rare and Precious Metals until his retirement last year, has been honored by the Dept. of the Interior with its Meritorious Service Award and Silver Medal. Mr. Keiser is now with the National Academy of Sciences in Washington, D. C.

Philip D. Bieser, who had been junior mining engineer with American

Make October Guest Month

Invite your associates to attend the October meeting of your Local Section.

Show him AIME in action.

Tell him of other AIME meetings he would be interested in attending. (Refer to page 1180 for coming events.)

Ask him to fill in an application blank.

Every mining man should be a member of SME of AIME.

Smelting & Refining Co., is now associated with National Lead Co., Tahawus, N. Y., as junior mining engineer.

John W. Donaldson, formerly research engineer for Quebec Metallurgical Industries Ltd., is now with Heath Steele Mines Ltd., Newcastle, N. B., Canada, as assistant mill superintendent.

Kenneth C. Vincent of Westvaco Mineral Products Div., Newark, Calif., has been transferred to the Pocatello, Idaho, office of the company, a division of Food Machinery & Chemical Corp.

Richard W. Flagg, formerly with Dorr-Oliver Inc., Westport, Conn., is now associated with Denver Equipment Co., Denver.

C. W. THOMAS

Conrad W. Thomas has opened a consulting office in the Bank of the Southwest Bldg., Houston, and will perform general mining consultation assignments. He has a broad background in mine examination and mineral exploration, having worked for Texas Gulf Sulphur Co. from 1951 to 1956 as a mining engineer. His work in that period took him all over Latin America and to Japan, Europe, North Africa, the Middle East, and many parts of the U. S. Recently he has been doing consulting work in Central America and southern and western U. S. for the Pan American Sulphur Co. and others.

P. N. Miles has been named manager of the mining sales division of Christensen Diamond Products Co., succeeding **Alan Conner** who is directing operations at Castle Concrete Co., a subsidiary of Christensen. Mr. Miles, who joined the company in 1951, has been concerned with the sale of diamond bits and associated equipment to the mining and construction industries. He is a graduate of the University of Utah.

Charles D. Rubert has resigned from Gannett, Fleming, Corddry and Carpenter Inc., Harrisburg, Pa., to join the staff of Barrett, Haentjens & Co.,

Hazleton, Pa., manufacturers of centrifugal pumps and accessories.

Philip L. Jones, consulting mining engineer, has been appointed director and consultant to the Mining and Metallurgical Div., Bruce Williams Laboratories, Joplin, Mo. He will continue independent work in his consulting office in Joplin.

R. C. H. Goldsmith has taken a position with the South American Gold and Platinum Co., Bogata, Colombia.

Au-ngse-Ho is now employed by the Strandberg Mines Inc., Anchorage, Alaska. He graduated in June with an M.S. in geological engineering from Montana School of Mines.

C. P. De FERARI

C. P. De Ferari, previously associated with Compania Minera Aguilar in Jujuy, Argentina, is now employed by the Consorcio Minero del Peru of Lima, Peru, as mine superintendent at the San Juan de Lucanas gold and silver mine.

S. Anderson, formerly general superintendent for Manganese Inc., Henderson, Nev., is now project manager, Lucky Mc uranium operation, Riverton, Wyo., for Utah Construction Co.

Derek J. Ottley has left Peru and has taken a position as lecturer in mineral dressing at the Royal School of Mines, London, England. He also plans to work for a Ph.D. degree in some practical aspects of mineral dressing.

J. C. Dotson, formerly associated with the College of Mines at the University of Idaho, is now at the College of Mines, the University of Arizona in Tucson.

A. D. Sisk is now associated with the U. S. Bureau of Mines, Div. of Health and Safety, Washington, D. C.

Phelps Dodge Corp., Douglas, Ariz., announced the following personnel changes in western operations: **Warren E. Fenzi,** who is presently general superintendent of the Morenci Branch, has been named assistant to general manager with offices in Douglas; **John A. Lentz, Jr.,** presently general superintendent of the New Cornelia Branch at Ajo, Ariz., has been transferred to Morenci, and has become general superintendent of the Morenci Branch; **Lyle M. Barker** is manager of operations at Morenci; **James A. Briggs** has become general superintendent of the New Cornelia Branch; and **Troy B. Hinton** has become mine superintendent, replacing Mr. Briggs.

Donald Jenkins has been appointed southeastern district manager for Atlas Copco Eastern Inc. with headquarters in Knoxville, Tenn. He had been general mine foreman, Ivanhoe Mine, New Jersey Zinc Co., Austinville, Va.

H. T. Urband, formerly lieutenant jg, USN, is now working as a geologist at Phelps Dodge Corp., Morenci, Ariz.

L. S. LOHR

Lewis S. Lohr has joined the staff of **Keith N. Meador,** consulting mining geologist in Reno, Nev., and is currently engaged in exploration and development of iron properties. He graduated recently from Mackay School of Mines in Reno.

Doring C. Dahl has left the North Carolina State Highway Dept. after working on core drilling of two proposed tunnels on the Interstate highway system. He was field geologist in charge of the program. Mr. Dahl is now associated with American Smelting and Refining Co., Page Mine, Kellogg, Idaho, as geologist and mining engineer.

Edward A. Krisher has been employed by the Dravo Corp., Shaft and Tunnel Dept., Pittsburgh, as an engineer.

Patrick V. Gallagher has been appointed vice president of McDowell Co. Inc. for its Dwight-Lloyd Div., Cleveland. He joined Dwight-Lloyd in 1955 as chief engineer. Since 1950 he had been assistant chief engineer of M. H. Treadwell Co., New York.

Robert W. Thomas, a member of the Trane Co., Los Angeles office, since 1950, has been promoted to manager of the firm's New York office. He joined Trane in 1949 shortly after receiving an engineering degree at the University of Southern California.

L. A. Quiñones is now employed by Cerro de Pasco Corp. as mine foreman in Peru.

T. G. GRANRYD

T. G. Granryd has been appointed manager of the Product Improvement Dept. of The Frank G. Hough Co., Libertyville, Ill., a subsidiary of International Harvester Co.

E. G. Bailey, former president of ASME, has been made an honorary member of the Society. Before his retirement he was vice president of Babcock and Wilcox.

Arthur Horen, formerly with U. S. Steel Corp., is now employed by Universal Atlas Cement Co. as director of the Raw Materials Div., New York. Universal Atlas is a U. S. Steel subsidiary.

Harry M. Callaway is now an analyst for the U. S. Bureau of Mines, Washington, D. C., working on analyses for the lead and zinc industries. He has been with the Eagle-Picher Co. as exploration geologist.

E. J. Perry, Eldorado Mining and Refining Ltd., has gone to Europe on an extended tour, and is planning a number of visits to mining operations and plants of various mining equipment manufacturers while abroad. Mr. Perry expects to return to Canada late this fall.

Donald J. LeGrand is now residing in Panama and is available as a consulting geologist.

H. E. Hawkes, previously with the Geology Dept., Massachusetts Institute of Technology, is now associated with Div. of Mineral Technology, University of California, Berkeley.

Earl M. Weaver has left Herb J. Hawthorne Inc. after ten years of

service. He is available as a consultant in Houston.

George E. McElroy has retired from the U. S. Bureau of Mines, Pittsburgh, after 43 years of government service and 40 years with USBM. He had been chief of the Ventilation Section since 1937.

J. P. Dempsey, previously associated with the Tripp Research Corp., is now employed by DELUSA, Dallas.

The Alumni Assn. of Lehigh University has given **William Henry Lesser** its highest award in recognition of his pioneering as a mechanical-electrical engineer in coal mine electrification, in development of the sand-flotation process for anthracite beneficiation, and in mine development in Greece and Mexico; for his leadership in professional engineering organizations; and for his service to the University.

R. G. ALLEN

Robert G. Allen has been elected vice president of Bucyrus-Erie Co., South Milwaukee. He had been president of Pesco Products and Wooster Divs., Borg-Warner Corp. Mr. Allen served two terms in Congress as the representative from the 28th district, Pennsylvania.

Nathan D. Adams has been named manager and **James M. Hamilton** assistant manager of the McIntyre Porcupine Mines Ltd. operation at Schumacher, Ont., Canada.

W. L. HUBER

W. L. Huber, manager, Diamond Tool Research Co. Inc., New York, has been elected president of the Industrial Diamond Assn. of America Inc. **Morris Winston,** Diamond Drill Carbon Co., New York, and **Don Wallace,** Wheel Trueing Tool Co., Detroit, were elected first and second vice presidents, respectively. The elections took place at the association's annual convention at St. Clair Shores, Mich.

L. F. Hickernell has been named vice president-engineering, Anaconda Wire & Cable Co. **Leonard L. Carter** succeeds him as chief engineer. **Henry V. Van Valkenburg** has been named general sales manager.

Justin L. Beeson has joined the engineering staff of Michigan Chemical Corp., and will be located at the company's research laboratories, St. Louis, Mich., where he will work on various engineering problems relating to rare earths. He had been a member of the technical staff of Goodyear Tire and Rubber Co., Akron, Ohio.

M. D. Hassialis, professor of mineral engineering and executive officer, School of Mines, Columbia University, New York, has entered into an arrangement with Haile Mines Inc. to develop new processes to treat low-grade refractory manganese ores, primarily for the steel industry.

Donald E. Bobo, Lancaster, Ohio, has been awarded the Kennecott Copper Fellowship of $2000 for two years of study at Harvard Business School, Cambridge, Mass. The scholarship is awarded annually by Kennecott Copper Corp., New York, to an engineer interested in the extractive and metal-using industries, and is awarded to candidates for the degree of master in business administration. Mr. Bobo graduated cum laude in mechanical engineering from Ohio University in 1955 and was recently separated from the U. S. Air Force.

Ian K. MacGregor, formerly general manager of Manning, Maxwell and Moore Inc., Stratford, Conn., has been appointed vice president-eastern operations, Climax Molybdenum Co., New York.

I. K. MacGREGOR

Paul L. Weller has been named eastern representative of the research

and development division of Spencer Chemical Co., Kansas City.

Mine Safety Appliances Co. has announced the appointment of **Allen M. Lundin** as sales engineer in the company's Minneapolis office. He has been associated with MSA since 1952, serving in the Billings, Mont., office.

R. J. MALLINA

R. J. Mallina has joined Gardner-Denver Co., Grand Haven, Mich., as research consultant. Prior to his retirement, he was development executive with Bell Telephone Laboratories.

George L. Innes is heading the newly combined chemical sales and development activities of Climax Molybdenum Co. He joined Climax in 1955 as manager of chemical sales.

A. L. Geisinger, vice president and general manager, Plastics Div., Diamond Alkali Co. since 1953, will retire at the end of the year. He is a veteran of 38 years with the company. He will be succeeded as manager by **James P. Okie,** assistant general manager.

George W. Kratz, financial vice president, Pittsburgh Consolidation Coal Co., has retired after 50 years with the company. He will continue as a member of the board of directors and of the executive committee.

J. Wilfred Spooner has been named eighth minister of mines, Dept. of Mines, Ontario Province, Ottawa. His home is in Timmins and is in the Porcupine district, one of Canada's largest gold mining areas.

Sidney L. Groff joined the staff of Montana Bureau of Mines and Geology, Butte, as geologist and head of the groundwater branch. He received B.S. and M.S. degrees from Montana State University, and has been attending the University of Utah.

Victor N. Antaki is now plant manager, Air Products Inc., West Palm Beach, Fla.

K. H. BAESSLER

Karl H. Baessler, who has been associated with The Colorado Fuel and Iron Corp. since 1934, has been appointed director of wire product development for the Colorado, Wickwire Spencer, and Pacific Coast Divs. of the company. He will continue to make his headquarters in Oakland, Calif., the CF&I plant there.

Howard M. Stewart, Ray Mines Div., Kennecott Copper Corp., Ray, Ariz., has been named secretary-treasurer of the Eastern Chapter of Arizona Soc. of Professional Engineers for 1957-1958. Others named include:

George W. Legters, Jr., Arizona State Highway Dept., Globe, is president of the Chapter, and **Starling H. Morse**, Gila County, Ariz., Engineer, Globe, is vice president. **C. Leroy Hoyt**, Ray Mines Div., Kennecott Copper Corp., Ray, Ariz., is chairman of the Public Relations Committee of the Arizona Society.

R. P. BROOKS

Robert P. Brooks, northwestern sales manager, Bucyrus-Erie Co., Seattle, has been promoted to assistant sales manager, large machines and blast hole drills, and is working from the South Milwaukee office.

Edward F. Hahnfeldt has been named Pittsburgh district sales manager for Mine Safety Appliances Co. He had been a member of the sales staff of Velocity Power Tool Co., Chicago, a subsidiary of MSA. The company also announced that **Donald F. West** has been transferred to Atlanta as sales manager. He had been serving as product line manager in MSA's Pittsburgh office.

R. H. Winters took office on September 9 as president of Rio Tinto Mining Co. of Canada, Ltd., Toronto. **J. N. V. Duncan**, managing director of the parent company in London, had been serving as president of the Canadian firm. He remains as a member of the board of directors.

H. R. COLEN

Henry R. Colen, formerly geological-mining engineer, International Minerals and Chemicals Corp., Columbia, Tenn., has been named general manager of Minquim Internacionales S.A. de C.V.'s new subsidiary, headquartered in Monterrey, Mexico.

Samuel Riker, Jr., treasurer of The New Jersey Zinc Co., New York, has also been elected secretary.

Ross E. Gamble, formerly mill superintendent, Copper Range Co., Freda, Mich., is now assistant mill superintendent, White Pine Copper Co., White Pine, Mich.

Harry L. Schell, mining engineer, has joined the health and safety staff, U. S. Bureau of Mines, Duluth.

G. Grassmuck has left Argentina to return to Canada where he will be working for Quebec Metal Mines Accident Prevention Assn., Bourlamaque, Que., Canada.

Dale E. Hirschfeld, sales engineer, Jeffrey Mfg. Co., has been transferred to the Bluefield, W. Va., Beckley district office.

Books

(Continued from page 1046)

pp., members—$3.25, nonmembers—$6.50, 1957.—This manual is a most complete and comprehensive foreman training aid. The manual contains 12 chapters covering every major aspect of occupational accident prevention. •

Mining Directory Of Minnesota, 1957, by Henry H. Wade and Mildred R. Alm, *Mines Experiment Station*, University of Minnesota, Minneapolis, 288 pp., $1.00.—A list of all operating companies and major holding organizations identified with Minnesota iron ranges, together with officials, subsidiary and affiliated companies, as well as pertinent information, general statistics, and maps.

Route-Mapping and Position-Finding in Unexplored Regions, by W. Filcher, E. Przybyllok, and T. Hagen, *Academic Press Inc.*, 111 Fifth Ave., New York 3, N. Y., 288 pp., $9.00, 1957.—Ground route mapping as distinguished from that making full use of photogrammetry is the main subject of this book. The two principal parts of the book are given over to a method of route mapping and position finding. The third part takes account of photography as an aid in route mapping and topographical surveys. Each part has its own author who is an expert in his field. •

Handbook for Uranium Prospectors Revised, by the *U. S. Atomic Energy Commission and the U. S. Geological Survey*, Washington 25, D. C., 75¢, 1957.—The revision includes an enlarged section on the geological occurrence of uranium based on the

large amount of new information developed during the growth of the uranium industry since the earlier edition (1951). The booklet is designed to answer inquiries received by Government agencies as a result of the AEC's domestic uranium procurement program.

Geological Field Methods, by Julien W. Low, *Harper & Brothers,* 489 pp., $6.00, 1957.—For the inexperienced man in the field, this manual contains, in addition to the major outlines of geologic field work, hints and help on the minor aspects of methods and procedures which the author, from his long experience, has found most often needed in carrying out assignments. •

Uranium and Thorium-Bearing Minerals in Placer Deposits in Idaho, by J. Hoover Mackin and Dwight L. Schmidt, *Idaho Bureau of Mines and Geology,* University of Idaho, Moscow, Idaho, 9 pp., 25¢, 1957.—It deals with the economic geology of one of the big and relatively undeveloped mineral resources, the uranium and thorium-bearing placer deposits of central Idaho.

Reactive Minerals In Idaho, by C. F. Cook, *Idaho Bureau of Mines and Geology,* University of Idaho, Moscow, Idaho, 5 pp., 25¢, 1957.—The pamphlet contains reports on uranium and thorium deposits in Idaho.

Some Occurrences of Uranium and Thorium in Montana, by Leonard D. Jarrard, *Montana Bureau of Mines and Geology,* Room 203-B, Main Hall, Montana School of Mines, Butte, Montana, 90 pp., $1.00, 1957.—The main reason behind the publication of this report is the increased number of inquiries concerning the status of known and potential uranium and thorium deposits in the state.

Hydrometallurgy of Base Metals, edited by George D. Van Arsdale, *McGraw-Hill Book Co. Inc.,* revised edition, 360 pp., $9.50, 1957.—Here is the practical reference guide that fully explains all steps in the recovery of base metals by leaching—from the preparatory ones of crushing, grinding, and roasting, through the leaching processes and the final precipitation of the metal. All kinds of information—operating data, flow diagrams, cost figures, equipment descriptions—are included to make this book as clear and useful a guide as possible. •

Nonmetallic Minerals, by Raymond B. Ladoo and W. M. Myers, *McGraw-Hill Book Co. Inc.,* revised edition, 605 pp., $11.00, 1957.—This book provides a comprehensive coverage of nonmetallic minerals essential as raw materials to the chemical, construction, ceramic, fertilizer, abrasive, and other industries. Whatever your question on a particular non-

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Books

(Continued from page 1171)

metallic mineral you can locate accurate dependable answers quickly in this comprehensive reference. •

Mine Plant Design, by W. W. Staley, *McGraw-Hill Book Co. Inc.*, revised edition, 540 pp., $10.00, 1957.—This book provides, in a single convenient volume, material which will be helpful to mine engineers, operators, and superintendents in choosing mine equipment, and in solving kindred problems of construction, machines, etc. •

Corporate Secretary's Manual and Guide, revised edition, *The Parker Publishing Co. Inc.*, Englewood Cliffs, N. J., 1600 pp., $12.50, 1957.—The manual outlines all the duties of all corporate officers and provides the corporate secretary with a convenient checklist of the things that must be done for the efficient and legally sound handling of corporate affairs. It shows how to prepare and conduct corporate meetings; explains proper parliamentary procedure; gives invaluable assistance on stocks, dividends, taxes, borrowing, and thousands of other matters with which the corporate officer must deal. •

Professional Engineers' Income and Salary Survey, by the *National Society of Professional Engineers*, 2029 K St., N. W., Washington 6, D. C., $1.00, 1957.—The survey, third in a series started by the National Society in 1952, is based on an analysis of 17,000 questionnaires returned from registered professional engineers in all the technical branches. The survey shows an increase of 21 pct in the median earnings of engineers during 1952-1956, as indicated on the questionnaires returned, and gives other valuable information.

Supplement to **Encyclopedia of American Associations**, published by the *Gale Research Co.*, 1116 Book Tower, Detroit 26, Mich., $15.00, 1957.—Nine hundred national associations, societies, organizations, and labor unions are listed in the supplement. Each listing includes the national headquarters, executive secretary, number of members, founding date, and a brief description of the activities and purpose.

Instructional Film Research Reports volume II, published by the *Office of Technical Services*, U. S. Dept. of Commerce, Washington 25, D. C., 1033 pp., $6.00, 1957.—Volume I of this series, PB 111000 **Instructional Film Research 1918-1950**, 185 pp., also is available from OTS at $2.50 a copy.

A Preliminary Report on the Stratigraphy of the Uranium-Bearing Rocks of the Karnes County Area, South Central Texas, by D. Hoye Eargle and John L. Snider, prepared in cooperation with the *USGS* by Bureau of Economic Geology, *The University of Texas*, Austin 12, Texas, 25 pp., 50¢, 1957.—This pamphlet presents results of a preliminary investigation of the stratigraphy of the uranium-bearing formations through Karnes County and adjoining counties to the southwest and of the correlation of the potentially uranium-producing beds in drill holes and surface exposures. Several contacts were mapped in reconnaissance in the area.

National Science Foundation publications are now available and lists will be sent upon request. Address inquiries to Publications Office, National Science Foundation, Washington 25, D. C.

Laws and Regulations Governing Mineral Rights in Arizona, by Victor H. Verity, revised edition, *Dept. of Mineral Resources*, State of Arizona, Phoenix, Ariz., 71 pp., 30¢, 1957.—This revised edition, made necessary by changes in law, is limited to the Federal and State mining laws as they apply within the state of Arizona. The department keeps informed on new and

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Books

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pending legislation, changes in freight rates, tariffs, trade agreements, taxations, and other factors which affect the mining industry.

U. S. Bureau of Mines Publications

Request free publications from:

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RI 5280 Some Variables in Treatment of Scheelite with Soda Ash.

RI 5282 Method of Fluorometric Determination of Beryllium.

RI 5283 Synthetic Mica Investigations. VIII—Manufacture of Fluor-Phlogopite by Internal Electric-Resistance Melting Process.

RI 5284 Leadville Drainage Tunnel Second Project, Lake County, Colo.

RI 5285 Mercury Deposits in Nevada and Malheur County, Ore.

RI 5286 Analysis of Crude Shale Oil. Part 2, Some Brazilian and U. S. Oils.

RI 5287 Anthrafines for Producing Metallurgical Coke.

RI 5288 Methane Buildup During Cutting and Continuous Mining Operations.

RI 5289 Waterborne Wastes of Mineral Industries, New England-New York Region.

RI 5290 Anthracite Mechanical-Mining—Use Yielding Steel Supports in Combination with Backfilling for Mining Thick Flat Beds.

RI 5291 Diamond Bit Performance in Schist.

RI 5293 Sunthetic Asbestos. III—Synthesis and Properties of Fibrous Potassium-Lead Silicate.

IC 7766 Mining Methods and Costs, Standard Uranium Corp., Big Buck Mine, San Juan County, Utah.

IC 7767 Potential of Heavy-Mineral-Bearing Alluvial Deposits in Pacific Northwest.

IC 7768 Sampling Deep Ore Deposits by Rotary Drilling and Methods of Surveying and Controlling Direction of Drill Holes.

IC 7770 Reconnaissance of Sulfur Resources in Wyoming, Colorado, Utah, New Mexico, Arizona.

IC 7774 Mining Methods and Costs at Westside Mine of Eagle-Picher Co., Cherokee County, Kan.

IC 7776 National Annual Diesel Fuel Survey, 1956.

RI 5292 Mining, Manganese Deposits in Maggie Canyon Area, Artillery Mts. region, Mohave County, Ariz.

RI 5294 Petrographic Composition and Free-Swelling Properties of Chilton Coal.

RI 5295 Expansion of Coal in Sole-Heated Oven. Dry Bulk Density, Moisture Content, Particle Size.

RI 5296 Tuffs near Lysite, Wyo., for Selenium.

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monia-Synthesis Gas from North Dakota Lignite in Annular-Retort Gasifier.

RI 5298 Volatilization of Tin Chlorides from Slime.

RI 5299 Relative Corrosion Resistance of Titanium and Some of Its Alloys.

RI 5300 Relationship of Composition to Thermal Stability in Huebnerite-Ferberite Series of Tungstates.

RI 5301 Mineral Dressing Study of Manganese Deposits of Batesville, Ark., District.

RI 5302 Low-Temperature Tar from Fluidized Carbonizing Reactors.

RI 5303 Bulk Sampling of Diamond Drilling, Dudley Manganese Deposit, Northern District, Aroostook County, Me.

RI 5305 Preparation of High-Purity Electrolytic Chromium.

RI 5307 Effects of Substituting Various High and Low Volatile Coals in Foundry Coke Blends Carbonized in Tuscaloosa and Sole-Heated Ovens.

RI 5308 Laboratory Cleaning Trials on Pennsylvania Anthracite in Baum-Type Jig and Dense-Medium Pilot Plant.

RI 5309 Characteristics, Analyses of 92 Colorado Crude Oils.

RI 5310 Anchorage Testing of Mine Roof Bolts. Part 3, Expansion-Type $\frac{5}{8}$-in. Bolts.

RI 5311 Consumable-Electrode Arc Melting of Titanium and Its Alloys.

RI 5313 Copper-Lead-Zinc Deposits, Cabarrus and Union Counties, N. C.

RI 5318 Development of Transistor-Type Telephone System for Mine Rescue Operations.

OBITUARIES

Philip Barnett Bucky

An Appreciation By
M. D. Hassialis

It is with deepest regret that my colleagues and I of the School of Mines of Columbia University are called upon to record the passing of Philip Barnett Bucky (Member 1921), who died on Aug. 8, 1957.

Born in Chicago in 1899 and educated at the University of Illinois and Pennsylvania State College, Philip B. Bucky came to Columbia as an assistant professor of mining in 1929. As a result of successive promotions he was appointed professor of mining in 1942. From 1946 to 1952 he served the School of Mines as its executive officer, a post which he relinquished owing to failing health.

Prof. Bucky served his country in both world wars; in the first, as a carpenters mate in the U. S. Navy (1918) and as a captain of the American Red Cross (1918-1919); in World War II as consultant to a number of governmental agencies, particularly the Bureau of Mines.

Although Prof. Bucky's professional achievements range widely over the field of mining, he will perhaps best be remembered for his contributions in and his pioneering of the field of barodynamics and the application of photoelastic methods to the study of stress distribution in underground mining structures. With the maturing knowledge in this field of investigation and with the deeper appreciation of the need to determine physical properties of rocks in situ, the value of his early work on laboratory specimens is sometimes lost sight of. However, such hindsight should not prevent us from recognizing and appreciating both the courage and the determination involved in working with admittedly crude approximations to the ultimate mining problem. The important thing is that a start has been made.

Prof. Bucky was a prolific writer—a statement attested by his long list of publications in the journals of AIME and in other journals, by his authorship of a book on *Mining by Block Caving*, and by his associate editorship of Peele's *Mining Engineers Handbook*. He was also the author of a number of patents on underground supports, mining methods, and crushing devices. His membership in various technological societies is too long to record but special mention should be made of

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Obituaries

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his membership in AIME, for this was an activity dear to his heart.

We at Columbia make public acknowledgment of our debt to his leadership and record herein our promise that the paths he has broken shall be trod by those who follow.

Worth B. Andrews (Member 1950) died on June 26, 1957. at the age of 75. He was born in Audubon, Minn., and was educated at the Michigan College of Mines from which he graduated in 1960. Mr. Andrews was mining engineer and oil operator for his own company in San Antonio, Texas.

George H. Garrey (Legion of Honor Member 1905) died at his home July 23,1957, shortly after his 82nd birthday. A native of Manitowoc, Wis., he attended Chicago University and Michigan College of Mines. Mr. Garrey had been employed by the U. S. Geological Survey and had been employed as general manager for the Reynolds-Morse Corp. in Denver, Colo. For many years he was a consultant in mining geology.

Stacy H. Hill (Member 1937), retired mining engineer and former district manager for Ingersoll-Rand Corp., died at his home on July 7, 1957 Mr. Hill was born in Mobile, Ill., and graduated from the Michigan School of Mines.

John E. Jones (Member 1919) died July 1, 1957, at his home in Benton, Ill. Born in Buckley, North Wales, in 1883, Mr. Jones graduated as a civil engineer from the Valparaiso University in 1910. He was appointed as an Illinois State mine inspector in 1915 and two years later joined the Old Ben Coal Corp. for the remainder of his career until his retirement in 1953. During World War II, Mr. Jones was mine safety consultant to the Secretary of the Dept. of the Interior and in 1946 was coal mines administrator safety engineer and consultant.

Necrology

Date Elected	Name	Date of Death
1956	Algernon N. Borszich	Unknown
1921	Philip B. Bucky	Aug. 8, 1957
1902	H. E. Crawford Legion of Honor	Sept. 23, 1952
1953	Clayton R. Genung	Sept. 25, 1955
1918	Arthur W. Gray	Apr. 4, 1957
1921	William Hart	December 1956
1903	Charles K. Leith Legion of Honor	Unknown
1902	Lesley McCreath Legion of Honor	Aug. 18, 1957
1952	Melvin H. Ott	February 1956
1939	Edward L. Ralston	July 9, 1957
1943	James R. Russell	Aug. 5, 1957
1949	R. S. Sherwin	July 2, 1957
1945	Eugene Staritzky	May 15, 1957
1899	J. B. Tyrrell Legion of Honor	Aug. 26, 1957
1949	George A. White	May 5, 1956
1920	George F. Zoffman	June 5, 1957

Russell L. Kelce (Member 1945), president of the Peabody Coal Co. and board chairman of the Chicago Great Western Railway, died on June 30, 1957. Born in Pittsburg, Kan., in 1897, Mr. Kelce attended school there until at an early age it was necessary for him to go to work in the coal pits. Gov. Phil M. Donnelly had recently appointed him to the Missouri Resources and Development Commission.

Walter C. Kerrigan (Member 1948) died July 16, 1957. Mr. Kerrigan was assistant to the president of the International Nickel Co. of Canada Ltd. and its U. S. subsidiary. He had been associated with the firm since 1930. Born in New York, he attended Rutgers and Columbia Universities. Prior to joining Inco., he was president and sales manager of U. S. Nickel Co, liquidated in 1929.

D. Ford McCormick (Member 1920), 69, died on July 14, 1957, after an illness of several months. Mr. McCormick was born at Del Rio, Texas, and graduated from the University of Texas in 1908 with a civil engineering degree. Two years later he received a degree in mining engineering at the Colorado School of Mines. He managed mines in Costa Rica and Cuba. From 1941 to 1946, Mr. McCormick managed quicksilver mines near Winnemucca, Nev.; he served as mining consultant in Oregon from 1946 until his death.

E. S. McCurdy (Member 1917) died on July 13, 1957, at the age of 69. A native of Ohio, Mr. McCurdy studied at Princeton and Columbia Universities. Most of Mr. McCurdy's career as a mining engineer was spent with the Utica Mining Co. in San Francisco. Prior to his retirement he had been a mining consultant.

Oliver J. Neslage (Member 1936), vice president, commercial sales, Joy Manufacturing Co., died in Pittsburgh on June 19, 1957. Born in St. Louis in 1893, he graduated

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from the University of Illinois and during World War I served in the Corps of Engineers. His entire business career was spent with the Joy Manufacturing Co. and its predecessor, the Sullivan Machinary Co., joining the latter in 1916.

Martin F. Tynan (Member 1935) died recently at his home in El Paso, Texas. Mr. Tynan was originally from New York where he had graduated from Columbia University in 1922 with a degree in mining engineering. He worked as a mining engineer in Chile and Mexico. In 1954, he accepted a position as assistant manager for American Smelting & Refining Co. in El Paso, the same firm for which he had worked in Mexico City.

Ira H. Wynne (Member 1954) died suddenly on July 11, 1957, while on a business trip to Duluth. Born in Pennsylvania, Mr. Wynne attended Curry College in Massachusetts where he graduated with a B. S. degree and the University of Pittsburgh where he received an E. M. degree. Mr. Wynne had spent many years working in South America and was currently associated with Reserve Mining Co. as a consulting engineer.

Roger William Straus (Member 1916) died on July 28, 1957, of a heart attack at his home in Liberty, N. Y. Mr. Straus had retired in April as board chairman of the American Smelting and Refining Co. with whom he had been associated since 1914. A Princeton University graduate, Mr. Straus served as a member of the U. S. delegation to the United Nations General Assembly in 1954 and he was currently chancellor of the State Board of Regents. Mr. Straus had been a founder and was active in the National Conference of Christians and Jews.

MEMBERSHIP

Proposed for Membership
Society of Mining Engineers of AIME

Total AIME membership on Aug. 31, 1957, was 28,134; in addition 3,143 Student Members were enrolled.

ADMISSIONS COMMITTEE

Robert Grogan, Chairman; Frank Ayer, Jack Bonardi, A. C. Brinker, Jack B. Graham, F. A. McGonigle, F. W. McQuiston, Jr., G. R. Spindler, L. P. Warriner.

The Institute desires to extend its privileges to every person to whom it can be of service, but does not desire as members persons who are unqualified. Institute members are urged to review this list as soon as possible and immediately to inform the Secretary's office if names of people are found who are known to be unqualified for AIME membership.

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John C. Hagen, Iron Mountain, Mo.
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Student to Junior

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Coming Events

Oct. 3-5, Seventh Annual Exploration Drilling Symposium, University of Minnesota, Center for Continuation Study, Minneapolis.

Oct. 6-9, AIME Society of Petroleum Engineers, fall meeting, Adolphus, Baker, and Statler-Hilton Hotels, Dallas.

Oct. 10-11, ASME-AIME Coal Div., Joint Solid Fuels Conference, Chateau Frontenac, Quebec.

Oct. 11, AIME St. Louis Section, fall field trip to Illinois-Kentucky Fluorspar District.

Oct. 15-19, AIME, Society of Mining Engineers Annual Meeting and Southeastern States Mining Conference, Hillsboro and Tampa Terrace Hotels, Tampa, Fla.

Oct. 17, AIME Utah Section. Speaker: Robert C. Mayer; subject: *Furnace Practices and Material Handling at Geneva Steel Co.*

Oct. 24-25, ECPD-EJC, general assembly, Hotel Statler, New York.

Oct. 30-Nov. 1, AIME Rocky Mountain Minerals Conference, Denver.

Nov. 8, AIME Pennsylvania-Anthracite Section, fall technical meeting, Wilkes-Barre, Pa.

Nov. 8-9, AIME Central Appalachian Section, West Virginia Mining Inst., joint meeting, Greenbrier Hotel, White Sulphur Springs, W. Va.

Nov. 11-14, Society of Exploration Geophysicists, 27th annual meeting, Statler-Hilton Hotel, Dallas.

Nov. 21, AIME Utah Section, panel discussion: *Mining Industry: New Methods, Machines, and Materials;* moderator—Norman L. Weiss; panel—James D. Moore, Joseph Rosenblatt, P. K. Richardson, Harvey Mathews. Salt Lake City.

Dec. 12, AIME Utah Section, *The Engineering Aspects of American Gilsonite Developments* by Roy E. Nelson; Salt Lake City.

Jan. 16, 1958, AIME Utah Section, panel discussion: *Industrial Engineering Practices in the Mineral Industries;* moderator—I. K. Hearn; Salt Lake City.

Feb. 16-20, 1958, AIME Annual Meeting, Hotels Statler and Sheraton-McAlpin, New York.

Apr. 17-19, AIME Pacific Northwest Regional Conference, Spokane.

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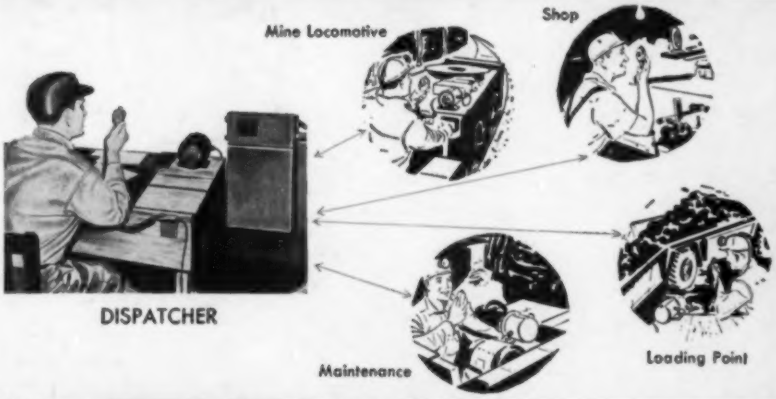

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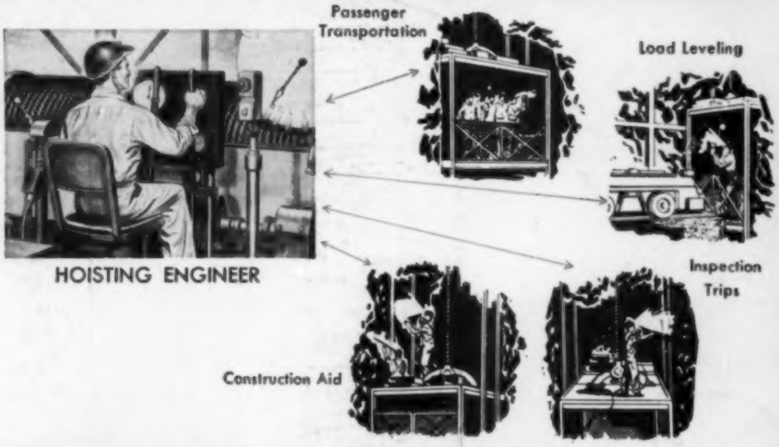

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